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I. C. 6220

DECEMBER, 1929

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

STATE REGULATIONS GOVERNING THE USE OF TROLLEY WIRES IN MINES

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BY

L. C. ILSLEY AND R. A. KEARNS

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

STATE REGULATIONS GOVERNING THE USE OF TROLLEY WIRES IN MINES[1]

By L. C. Ilsley[2] and R. A. Kearns[3]

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THE HAZARD OF TROLLEY WIRES

State safety codes give more attention to the question of trolley-wire installation than to any other phase of mine regulations pertaining to the use of electrical equipment. This attitude on the part of legislators is to be expected because trolley circuits offer a greater hazard than other circuits common to electrical installations in mines. In fact, this danger is so great that in mines outside of the United States the requirements either prohibit or greatly curtail the use of trolley circuits.

The three major hazards presented by a trolley circuit are (1) shock to persons, (2) the igniting of mine explosions, and (3) the igniting of mine fires. Most of the regulations are intended to lessen the first danger.

Shock Hazard

Of all circuits in mines, trolley circuits offer the greatest shock hazard. Most other circuits can be provided with a certain amount of insulation which prevents a person from coming in contact with the bare wire, but trolley circuits must be bare for their entire length. Most other circuits can be installed in such a manner as to prevent contact with them, but trolley wires must follow the alignment of the track, and usually because of the lack of head room they are so low that they offer a special hazard. In following the track, trolley circuits must pass under good and bad roof and encounter wet or dry conditions, as the case may be.

Some of the adverse conditions just mentioned can be modified, but generally at a considerable expense to the operator. Narrow roads can be widened to give clearance along the roadway on the side opposite trolley circuits. Bad roof can be taken down and additional supports placed, and where the roof is very bad, an arched support can be used. Top or bottom rock can be removed to make greater height. Drainage conditions can be improved. All of these changes which

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1 - The Bureau of Mines will welcome reprinting of this paper, provided that the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Circular 6220."

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make for a good safe trolley installation add to the cost of mining and could be carried to a point that would make this cost prohibitive.

A high proportion of the total shock accidents caused by electricity in mines comes from contact with trolley circuits. In some States the percentage of fatal accidents caused by contact with trolley wires is high, and in all States it is greater than necessary were proper precautions observed.

Gas and Dust Ignition Hazard

The hazard most to be feared from trolley wires is in connection with the initiation of major disasters. The arc from a 250 or 500 volt trolley wheel is most certain to set off gas if an explosive mixture is present at the point where such arcing takes place. The trolley wire with its feeder circuits frequently causes fires and offers a real gas and dust ignition hazard.

Track return is one of the greatest objections to the trolley circuit because it almost always uses a ground. This return often presents a hidden and unexpected danger due to the wandering of the return current in finding the best path back to the generators. Improper bonding of the track or no bonding may force the return current beyond the limits of the trolley-wire installation, even to the face workings where arcing at unbonded joints offers a serious hazard to the safety of all in the mine. Such cases have been called to the bureau's attention.

Fire Hazard

Many disastrous mine fires have been attributed to trolley circuits. This applies to metal mines particularly, although such fires have taken place in coal mines.

STATE REGULATIONS

Table 1 shows that a number of different States have tried to help the situation by the enactment of regulations governing the use of trolley circuits:

Discussion of Regulations

Item 1: Trolley wires to be shown on plan map.-- Seven States insist on this. There are two vital safety reasons for this requirement. First, the management shall at all times be informed how near the working areas the trolley circuits are being installed and shall take the responsibility for the liability of gas ignition at the face workings of gassy mines. Second, when there is a disaster, those in charge of rescue work may know from the map the position of all electrical equipment, including line wiring.

Item 2: Hangers to be efficiently insulated.-- Eight States have this regulation. From the standpoint of safety the rule is needed to minimize the fire hazard; such a requirement is also desirable from the standpoint of operation.

Item 3: Hangers properly spaced to prevent sag.-- This rule is covered by the safety codes of 10 States. It is necessary for both safety and efficiency.

Table[1] .- Rules relative to the use of trolley wire in mines.

States	Trolley wire to be shown on plan map	Hangers to be efficiently insulated	Hangers to be properly placed to prevent sag	Trolley wires to be suitably supported	Sag between supports "limited" to	Trolley to be installed on one side of roadway	Trolley wire to be shielded	Trolley to be maintained	Rule for termination of trolley wire	Forbidden to use trolley in working rooms	Power or lighting wires or both on same side as trolley wire	Trolley installed outside of rail	Operating locomotives in gassy section of mine prohibited	Lighting circuits fastened to drilled trolley ear	Voltage on trolley circuit limited	Means of cutting off power on branch circuits specified	Method of shielding or guarding suggested	Specific requirements for surface circuits	Stricter requirements for future installations	Height above rail or floor a safety factor	Size of trolley wire specified	Indirect mention of trolley wire	Illumination requirements for shielded points	Trolley wire to be sectionalized
Alebema							Yes													Yes				
Arizone																			Yes	Yes				
Californie (ell Mines)						Yes	Yes					Yes			Yes			Yes		Yes				
Californie(mines other than coal)	Yes	Yes		Yes	3 in	Yes	Yes				Yes	Yes			Yes		Yes	Yes	Yes	Yes		Yes	Yes	
Colorado (coal)	Yes						Yes										Yes					Yes		
Colorado (metel)			Yas		3 in	Yes	Yes					6 in			Yes	Yas				Yes				Yas
Idaho							Yes										Yes							
Illinois (coal)							Yes								Yes									
Illinois (metal)	Yes	Yes	Yes	Yes	3 in	Yes	Yes				Yes			Yes			Yes			Yes		Yes	Yes	
Indiena		Yes	[2]Yaa	Yes		Yes	Yes	Yes				5 in				Yes	Yas							
Iowa																						Yes		
Kansas																								
Kentucky		Yes	Yes	Yes			[3]Yes				Yes								Yes	Yes				
Maryland				Yes			Yes																	
Michigan																						Yes		
New Jersey																			Yes	Yes				
New York (mines other than coal)	Yes	Yes	Yes	Yes	3 in		Yes										Yes		Yes			Yes	Yes	
Ohio		Yes	Yes	Yes		Yes	Yes		Yes	Yes	Yes	6 in			Yes				Yes					
Pennsylvania (bituminous)	Yes	Yes	Yes	Yes	3 in	Yes	Yes				Yes		Yes	Yes	Yes	Yes	Yes			Yes		Yes		
Texas			Yes	Yes		Yes	Yes	Yes				6 in							Yes	Yes				
Utah (coal)	Yes	Yes	Yes	Yes		Yes	Yes		Yes		Yes		Yes				Yes		Yes	Yes		Yes		
Uteh (metel)	Yas	Yes	Yas	Yes	3 in	Yes	Yes				Yes						Yes		Yes	Yes		Yes	Yes	
Virginia							Yes								Yes		Yes			Yes				
Washington			Yas	Yas	3 in		Yas						Yes		Yes	Yes	Yes		Yas	Yas		Yas		
West Virginie							Yes				Yes		Yes				Yea			Yes				
Wyoming						Yes	Yes				Yes						Yes							

1.- At ell pleces less then 6½ feet above reil.

2.- Wire to ba kept taut.

3.- At landing or partings where trolley is less then 6½ faet above rail.

A sagging line is dangerous to the motorman, as it is constantly arcing and difficult for the trolley wheel to follow. Good workmanship tends to better operation and increased production.

Item 4: Trolley wire to be suitably supported.--This item is included in 12 codes of 11 States. It is a sensible requirement which tends to safer and more satisfactory operation. The constant operation of the trolley wheel under considerable tension requires sturdy supports. Slack and fallen trolley lines are dangerous not only to the motorman but to all other men working near the lines.

Item 5: Sag between supports limited.--This regulation is included in seven State codes and is a specific requirement which, if followed, will insure that the lines are taut. A limit of 3 inches is specified in all cases.

Item 6: Trolley to be installed on one side of roadway.--Nine States and 11 codes include this requirement. This is an essential safety measure, especially where men must travel along haulage roads. Ohio is still more specific in keeping the trolley wire on the side opposite room necks and refuge holes.

Item 7: Trolley wire to be shielded.--This ruling is strictly a safety requirement, and is covered in 21 codes representing the regulations of 18 States. There has been a general agreement, and rightly so, that when trolley wires are less than 6-1/2 feet from the rail, they should be shielded to prevent accidental contact with persons passing under or alongside them, especially at points where men have to expose themselves to this hazard. Undoubtedly many fatal accidents from trolley circuits might have been avoided had this regulation been strictly enforced.

Item 8: Trolley wire to be maintained.--This item, which would seem essential to good practice, has been generally overlooked and is covered by only two States. It is just as essential to maintain electrical equipment in good condition day by day as it is to have it properly installed in the beginning.

Item 9: Rule for termination of trolley wire.--This item is covered by Ohio, which forbids carrying trolley wire into any entry beyond the outside corner of the last breakthrough. Utah also covers this item in a rule requiring that trolley wire be kept back of the last breakthrough and at least 50 feet back of the first room where pillars are being drawn. Every State ought to have a rule governing the termination of trolley circuits in gassy mines.

Item 10: Forbidden to install trolley wire in working room.--Ohio covers this item in a rule which states that "No trolley wire shall be extended into or maintained in any room while being used as a working place."

Alabama perhaps covers this item, indirectly for gassy rooms, under the rule which states that "Trolley wires are prohibited in any part of the mine where safety lamps are used." The probable intent of this rule could undoubtedly be greatly improved by rewording. As the wording now stands, there might be a tendency to keep safety lamps out of mines where they are badly needed in order that trolley circuits might remain. Rooms should not be permanently wired.

Item 11: Power or lighting wires or both on same side as trolley wire.--This safety requirement, intended to insure one side of the roadway for save travel as far as electric shock is concerned, is a part of nine different codes affecting the laws of eight States.

Item 12: Trolley installed outside of rail.--Five States have deemed it necessary to enact this ruling which tends to safeguard the motorman by removing the trolley to one side. The wire to one side tends to prevent low-hanging wire from coming in contact with loaded cars.

Item 13: Operating locomotive in gassy section of mine prohibited.--This item is included in the regulations of four States. The intent of this rule is to avoid possible gas ignition from the arcing trolley wire. It is another way of stating where the trolley circuit should be installed. If there is no trolley wire, then there can not possibly be an arc from the improper use of the trolley locomotive.

Item 14: Lighting circuits fastened by means of drilled trolley ear.--This rule, which is covered by two States, insures good installation practice and a certain measure of increased safety.

Item 15: Voltage on trolley circuit limited.--This is primarily a safety measure, and from the standpoint of shock hazard it is an important one. It is quite possible that were all States to start anew, no pressure in excess of 250 volts would be permitted for operation of trolley circuits. There seems to be a strong feeling that voltage above 250 adds too great a shock hazard, and several States have legislated against the use of higher voltage. Seven States mention the question of voltage in their safety codes.

Item 16: Means of cutting off power on branch circuits specified.--In the maintenance of circuits in mines, men should not be called upon to work on live conductors. This item, which is a part of the code of four States, permits the isolation of branch circuit when work is to be done on any part of it.

Item 17: Method of shielding or guarding suggested.--Not only have a number of States insisted that trolley circuits shall be guarded, but 12 States also give suggestions as to how this should be accomplished. This shows how dangerous trolley circuits are considered, when looked upon as a personal hazard.

Item 18: Specific requirements for surface circuits.--Most of the rules apply to trolley circuits installed underground. However, two States, California and Utah, have specific regulations for surface trolley circuits about mines, which if not well installed, may offer a real shock hazard.

Item 19: Stricter requirements for future installations.--The general feeling of uneasiness in regard to trolley circuits is reflected in the fact that nine States have made more strict regulations to govern new installations than those that already exist covering circuits already installed. There is undoubtedly room for further work along this line.

Item 20: Height above floor or rail a safety factor.--The fact that 13 States have rules governing the height of the trolley wire above the floor or rail

again emphasizes the generally recognized danger of trolley wire as it exposes the workman to shock hazard in his daily routine work. It must be well above the height of the average man to remove this menace.

Item 21: Size of trolley wire specified.--In no Code is the size of trolley wire specified. From the standpoint of mechanical strength, wire below 2/0 should not be used. From the standpoint of operating voltage, 4/0 should probably be used in most 250-volt installations.

Item 22: Indirect mention of trolley wire.--This item was included to cover those cases where the trolley circuit was only incidentally mentioned; the rule has little if any bearing on the safe installation or maintenance of the trolley circuit itself.

Item 23: Illumination requirements for shielded points.--As a natural consequence of the rule for shielding wire at points where men have to work under or near the wire, the additional requirement is made that such points shall be adequately illuminated. This rule is included in codes of four States.

Item 24: Trolley wire to be sectionalized.--This is a requirement of Colorado's safety code and one that might well be in any safety code. Such sectionalization not only is a safety measure for men working on defective lines, but often will assist the operator in keeping a portion of his mine in operation while the damaged part is under repair. Lines should be sectionalized at least every 2,500 feet.

CONCLUSIONS

Twenty-six safety codes have been studied and about 175 rules have been considered in Table 1. In gathering together these data it is possible that other rules having application may have been overlooked. However, it is believed that most of the regulations dealing specifically with trolley circuits have been included.

In making a study of this kind one is impressed with the general recognition that is given to this hazard from the standpoints both of shock and ignition. As an advocate of safety, the U. S. Bureau of Mines can not encourage the use of trolley circuits in any mine if some other practical substitute can be used. In this connection the bureau wishes to call attention to the so-called "wireless mine" in which the operator endeavors to have no permanent wiring, using portable battery power for furnishing the electrical energy at the working face. This problem is covered fullin in Cooperative Bulletin 42 (C. W. Owings and D. C. Jones: Efficiency, Cost, and Safety of Storage-Battery Equipment in Bituminous Coal Mines, and some Comparisons with Wired Transmission of Power) of the U. S. Bureau of Mines and Carnegie Institute of Technology, published in 1929.

Composite Code for Trolley Circuits

The Pennsylvania Bituminous Mine Law, which includes one of the most comprehensive sets of regulations concerning trolley circuits, has been selected as a framework upon which to build a code containing the essential requirements of all the codes considered. The composite code follows:

1. A plan shall be kept at the mine, showing the location of all stationary electrical apparatus in connection with the mine electrical system, including permanent cables, conductors, lights, switches, and trolley lines . . .

2. In underground roads the trolley wires shall be installed as far to one side of the passageway as is practicable, and securely supported upon hangers, efficiently insulated, and placed at such intervals that the sag between points of support shall not exceed 3 inches.

3. All other wires, except telephone, shot-firing, and signal wires, shall be on the same side of the road as the trolley wires.

4. At all landings and partings where men are required regularly to work or pass under trolley or other bare power wires which are placed less than 6½ feet above top rail, a suitable protection shall be provided. This protection may consist of channeling the roof, placing boards along the wire, so as to extend below it, or the use of other approved safety devices.

5. All branch trolley lines shall be fitted with an automatic trolley switch or section insulator and line switch, or with some other device that will allow the current to be shut off from such branch headings.

6. Where wires for electric incandescent lamps are connected to the trolley circuit, the trolley ear to which connection is made shall be drilled to receive the lighting wires and provided with a set screw for securing the wires in place. Lighting wires shall not be wrapped or tied about the stems or studs of trolley hangers. The ground connection for lighting wires taken off the trolley circuit must be made to the track circuit.

7. All trolley and positive feed wires shall be placed on the opposite side of the track from refuge holes or necks of rooms.

8. No trolley wire shall be extended into or maintained in any room while it being used as a working place; no trolley or feed wire shall be extended into any entry beyond the outside corner of the last break-through. Trolley wires must be kept back of the last breakthrough and at least 50 feet back of the first room where pillars are being drawn.

9. Trolley wires shall be hung and kept not less than 5½ feet above the rail and shall be securely fastened and not permitted to sag.

10. All trolley wires shall be carried at least 6 inches outside of and parallel with the track rail.

11. Electric haulage by locomotives operated from a trolley wire is not permissible in any gassy portions of mines, except upon the intake air, fresh from the outside.

12. Trolley wires or other exposed electrical wires shall not carry a pressure above 275 volts.

13. In new installations the wire shall be shielded unless it is at least 7 feet above the top of the rail.

14. Underground trolley lines should be sectionalized by placing at every thousand feet in the line a switch which can entirely disconnect the line from the power supply.

15. Overhead lines above ground: Overhead transmission lines between the generating station or substation and the mine entrance shall be supported upon insulators, which shall be adequate in quality, size, and design for the voltage transmitted. Where such a line is more than 500 feet long, lightning arresters shall be installed in connection therewith at the entrance to the mine. Such line except in the case of trolley wires, shall be maintained not less than 14 feet above the ground at the lowest point, except at the point of entrance to the mine.

Mining Laws Reviewed in Survey of Regulations Pertaining to Trolley Wires

Mining Law of the State of Alabama		1928
Mining Code of the State of Arizona		1921
Electrical Safety Orders of California		1925
Mine Safety Orders of California (Mineral Mines Other than Coal)	Effective January 1,	1921
Coal Mining Laws of Colorado		1925
Bulletin No. 9, Mine Safety Standards (Metal Mines)	June 1,	1920
Mining Laws of the State of Idaho	Effective May 8,	1919
Laws and Regulations Governing the Operation of Metal Mines in Illinois	Effective January 1,	1922
Mining Laws of the State of Illinois		1927
Mining Laws of Indiana	In force,	1927
Mining Law of Iowa	Code of	1924
Kansas Mining Laws	May,	1918
Mining Laws of the State of Kentucky		1920
Mining Laws of the State of Maryland		1923
Mining Laws of Michigan		1913
Labor Laws of New Jersey		1921
Rules Relating to Mines (Mineral - Other than Coal- Mines of New York)		1918
Mining Laws of Ohio		1927
Bituminous Mining Laws of Pennsylvania		1921
Coal Mining Laws of Texas		1916
General Coal Mine Safety Orders (Utah)	Effective September,	1920
	Amended April 8,	1924
General Safety Orders Covering Underground Metal Mining Operations of Utah	Effective August 15,	1919
Mining Laws of Virginia	Effective June 16,	1924
Coal Mining Laws of Washington, as amended	1919 and	1927
Mining Laws of the State of West Virginia	Effective July 21,	1925
Coal Mining Laws of the State of Wyoming		1919

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

LIST OF MANUFACTURERS OF PERMISSIBLE EQUIPMENT AND EXPLOSION-PROOF ACCESSORIES, AS OF JANUARY 1, 1930[1]

By L. C. Ilsley[2]

More and more interest is being manifested in the subject of approved equipment for use in gassy and dusty mines. The United States Bureau of Mines receives numerous inquiries as to where accessories available for permissible applications can be obtained. The number of manufacturers holding approvals for complete outfits is constantly increasing. This circular has been prepared to present information on the subject in a way that should make it easily available. Anyone desiring more detailed information should ask for the current list of Bureau of Mines approved mine equipment, which is issued every six months.

MANUFACTURERS HOLDING APPROVALS

Air Compressors:

General Electric Co., East Lake Road, Erie, Pa.
Ingersoll-Rand Co., Painted Post, N. Y.
Sullivan Machinery Co., Claremont, N. H.

Loading Machines and Conveyors:

Bethlehem Steel Co., Elizabeth, N. J.
Bird Coal Co., Barnesboro, Pa.
Brown-Fayro Co., 940 Ash Street, Johnstown, Pa.
Chicago Automatic Conveyor Co., 1845 So. 55th Ave., Cicero, Ill.
Duncan Foundry & Machine Works, Alton, Ill.
Fairfield Engineering Co., Marion, Ohio.
Goodman Manufacturing Co., Halsted St. & 48th Place, Chicago, Ill.
Joy Manufacturing Co., Franklin, Pa.

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2 - Electrical engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

Lorain Steel Co., Johnstown, Pa.
Mancha Storage Battery Locomotive Co., 1909 South Kingshighway Blvd., St. Louis, Mo.
C. H. McCullough Engineering Co., Oliver Bldg., Pittsburgh, Pa.
Myers-Whaley Co., Inc., Knoxville, Tenn.
Oldroyd Machine Co., Cincinnati, Ohio.
South Fork Foundry & Machine Co., South Fork, Pa.
Sullivan Machinery Co., Claremont, N. H.
The Jeffrey Manufacturing Co., Columbus, Ohio.
The Northern Conveyor & Manufacturing Co., Janesville, Wis.
B. P. Tracy Co., 919 Fulton St., Pittsburgh, Pa.

Drills and Drilling Machines:

Chicago Pneumatic Tool Co., 1241 East 49th St., Cleveland, Ohio.
Martin-Hardsocg Co., 922 South Ave., Pittsburgh, Pa.
Ohio Brass Co., Mansfield, Ohio.
Sullivan Machinery Co., Claremont, N. H.
The Jeffrey Manufacturing Co., Columbus, O.

Coal-Cutting Machines:

Goodman Manufacturing Co., Halsted St., at 48th Place, Chicago, Ill.
Joy Manufacturing Co., Franklin, Pa.
Oldroyd Machine Co., Cincinnati, Ohio.
Sullivan Machinery Co., Claremont, N. H.
The Jeffrey Manufacturing Co., Columbus, Ohio.

Room Hoists:

Brown-Fayro Co., 940 Ash St., Johnstown, Pa.
South Fork Foundry & Machine Co., South Fork, Pa.
Sullivan Machinery Co., Claremont, N. H.

Mine Pumps:

Dravo-Doyle Co., Dravo Building, Pittsburgh, Pa.
General Electric Co., Schenectady, N. Y.
Goodman Manufacturing Co., Halsted St. & 48th Place, Chicago, Ill.
Westinghouse Elec. & Mfg. Co., East Pittsburgh, Pa.

Concrete Mixers:

Westinghouse Elec. & Mfg. Co., East Pittsburgh, Pa.

Rock-Dusting Machines:

American Mine Door Co., Canton, Ohio.
Diamond Machine Co., Monongahela, Pa.
Mine Safety Appliances Co., Pittsburgh, Pa.

Electric Switches and Junction Boxes:

Ohio Brass Co., Mansfield, Ohio.
Sullivan Machinery Co., Claremont, N. H.

Storage-Battery Locomotives:

Atlas Car & Manufacturing Co., Cleveland, Ohio.
General Electric Co., East Lake Road, Erie, Pa.
Goodman Manufacturing Co., Halsted St. & 48th Place, Chicago, Ill.
Ironton Engine Co., Ironton, Ohio.
Mancha Storage Battery Locomotive Co., 1909 South Kingshighway Blvd., St. Louis, Mo.
The Jeffrey Manufacturing Co., Columbus, Ohio.
Westinghouse Elec. & Mfg. Co., East Pittsburgh, Pa.
Geo. D. Whitcomb Co., Rochelle, Ill.

Storage-Battery Power-Trucks:

General Electric Co., East Lake Road, Erie, Pa.
Mancha Storage Battery Locomotive Co., 1909 South Kingshighway Blvd., St. Louis, Mo.
The Jeffrey Manufacturing Co., Columbus, Ohio.
Westinghouse Elec. & Mfg. Co., East Pittsburgh, Pa.

Mine Telephones:

Western Electric Co., Inc., New York City.

Electric Lamps:

Concordia Electric Co., 200 East Ohio St., Pittsburgh, Pa.
Edison Storage Battery Co., Orange, N. J.
Koehler Manufacturing Co., Marlboro, Mass.
Mine Safety Appliances Co., Pittsburgh, Pa.
National Carbon Co., Inc., Frelinghuysen Ave., Newark, N. J.
Witherbee Lamp Co., Springfield, Mass.

Flame Safety Lamps:

Koehler Manufacturing Co., Marlboro, Mass.
Wolf Safety Lamp Co., of America, Inc., 68-72 Emerson Place, Brooklyn, N. Y.

Methane Indicators and Detectors:

Mine Safety Appliances Co., Pittsburgh, Pa.
Oxweld Acetylene Co., Carbide & Carbon Bldg., 30 East Forty-second Street, New York, N. Y.
F. O. Willhofft, consulting engineer, 11 West 42nd Street, New York, N. Y.
Wolf Safety Lamp Co., of America, Inc., 68-72 Emerson Place, Brooklyn, N. Y.

Single-Shot Blasting Units:

Concordia Electric Co., 200 East Ohio St., Pittsburgh, Pa.
Davis Instrument Mfg. Co., Inc., 513 East 36th Street, Baltimore, Md.
E. I. Du Pont de Nemours & Co., Wilmington, Del.
Mine Safety Appliances Co., Pittsburgh, Pa.
National Carbon Co., Inc., Frelinghuysen Ave., Newark, N. J.

MANUFACTURERS OF ACCESSORIES FOR PERMISSIBLE OUTFITS

Explosion-Proof Type Motors:

The following companies manufacture explosion-proof type motors. Such motors as have already passed prescribed tests can be used on permissible applications without the necessity of further tests by the Bureau of Mines, in so far as the motors are concerned:

Continental Electric Co., Inc., 323 Ferry St., Newark, N. J.
Crocker-Wheeler Electric Mfg. Co., Ampere, N. J.
General Electric Co., Lynn, Mass.
Goodman Manufacturing Co., Halsted St. & 48th Place, Chicago, Ill.
Louis Allis Co., 133 Stewart St., Milwaukee, Wis.
South Fork Foundry & Machine Co., South Fork, Pa.
Star Electric Motor Co., Miller St., N.J.R.R. Ave. & King St., Newark, N. J.
The Jeffrey Mfg. Co., Columbus, Ohio.
The Reliance Electric & Engineering Co., 1042-1080 Ivanhoe Road, Cleveland, Ohio.
Westinghouse Elec. & Mfg. Co., East Pittsburgh, Pa.

Explosion-Proof Type Switch Gear:

The following companies manufacture explosion-proof type switch gear. Such switch gear as have already passed prescribed tests can be used on permissible applications without the necessity of further tests by the Bureau of Mines, in so far as the switch gear is concerned.

Clark Controller Co., Cleveland, Ohio.
Crocker-Wheeler Electric Mfg. Co., Ampere, N. J.
General Electric Co., Schenectady, N. Y.
Goodman Manufacturing Co., Halsted St. & 48th Place, Chicago, Ill.
Ohio Brass Co., Mansfield, Ohio.
The Jeffrey Manufacturing Co., Columbus, Ohio.
Union Electric Manufacturing Co., 926 Juneau Ave., Milwaukee, Wis.
Ward-Leonard, Mt. Vernon, N. Y.
Westinghouse Elec. & Mfg. Co., East Pittsburgh, Pa.
Whitman & Co., Punxsutawney, Pa.

Storage Batteries for Permissible Outfits:

The following companies have developed storage batteries which have been judged by the Bureau of Mines to be suitable for use on permissible-type battery outfits. However, the design of the battery box is always very important and must be considered by the bureau before it approves any given type of battery

Edison Storage Battery Co., Orange, N. J.
Electric Storage Battery Co., Allegheny Ave. & 19th St., Philadelphia, Pa.
Gould Storage Battery Co., Inc., Depew, N. Y.
K. W. Battery Co., Chicago, Ill.
Philadelphia Storage Battery Co., Ontario & C Streets, Philadelphia, Pa.

Explosion-Proof Type Headlights:

The following companies manufacture explosion-proof type headlights. Such headlights as have already passed prescribed tests can be used on permissible applications without the necessity of further tests by the Bureau of Mines, in so far as the headlights are concerned.

General Electric Co., Erie, Pa.
Goodman Manufacturing Co., Halsted St. & 48th Place, Chicago, Ill.
Mancha Storage Battery Locomotive Co., 1909 South Kingshighway Blvd., St. Louis, Mo.
The Jeffrey Manufacturing Co., Columbus, Ohio.
Westinghouse Elec. & Mfg. Co., East Pittsburgh, Pa.

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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

BARITE AND BARIUM PRODUCTS

PART II. BARIUM PRODUCTS

BY

R. M. SANTMYERS

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

BARITE AND BARIUM PRODUCTS[1]

Part II. Barium Products

By R. M. Santmyers[2]

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1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6223."

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INTRODUCTION

Barium is derived from two minerals, barite (barium sulphate) and witherite (barium carbonate). Metallic barium is at present of little commercial importance, although small quantities are used as a scavenger of gases in the manufacture of radio tubes and as a hardening agent for lead; however, barium compounds, many of which are manufactured from the minerals by chemical processes, constitute an important class of commercial products and enter into both domestic and foreign trade in considerable quantities.

Barium products may be roughly divided into three classes: (1) Ground barite; (2) lithopone; and (3) barium chemicals. Ground barite - crude ore which has been washed, jigged, and ground very fine - is used as a filler in numerous substances, such as paper, paints, rubber, etc., where a heavy, chemically inert substance is required. Lithopone - a mixture of approximately 70 per cent barium sulphate, 26 to 28 per cent of zinc sulphide and 1 to 3 per cent of zinc oxide - is a widely used white pigment. Barium chemicals find employment in a wide variety of industries. The manufacture of lithopone, at present, consumes by far the larger part of the crude barite produced and imported into this country and has expanded rapidly in recent years. Prior to the war the entire lithopone industry of the United States was located along the Atlantic coast practically at tidewater. The barite required by the industry was supplied by imports, chiefly from Germany, as the imported material cost less than domestic barite. War conditions forced lithopone manufacturers to develop deposits in Georgia, Tennessee, South Carolina, and elsewhere in the Southern States in order to obtain supplies of raw material. At the same time the manufacture of lithopone was started in the Middle West, utilizing crude barite from Missouri. With the termination of the war and the reappearance of German barite on the American market, the Southern field is finding it more and more difficult to furnish the eastern lithopone industry with crude ore.

The list of commercial barium chemicals includes blanc fixe or artificial sulphate of barium, barium carbonate, chloride, nitrate, monoxide, dioxide, hydroxide, sulphide, chlorate, and chromate. A few of these chemicals, however, are of only small importance. Barium sulphide (an intermediate product from which the majority of the other barium chemicals are made), blanc fixe and barium dioxide, chloride, nitrate, carbonate and hydroxide are the barium chemicals that are used extensively.

Prior to the war the United States was wholly dependent on imports for its supply of barium chemicals. Germany was the largest producer and furnished about two-thirds of the domestic requirements. Between 80 and 90 per cent of the imports were represented by three barium salts - barium carbonate, barium chloride, and barium dioxide. The cessation of imports and the war demand for barium chemicals (barium nitrate, blanc fixe, and barium dioxide) resulted in the establishment of a barium chemical industry in this country.

Domestic plants were erected chiefly in the East and Southeast, although there were plants in Ohio and Illinois. In some cases the plants were located relatively close to the domestic deposits of barite. The chief markets for barium chemicals were in the East, and upon the termination of the war domestic products came into direct competition with imported chemicals. The eastern plants are so situated that they can use either imported or domestic barite as raw material, and therefore are able to compete with imported barium chemicals.

Those plants located close to the southern barite deposits and those in the Middle West depend solely on domestic crude barite which costs more than that imported from Germany. In these areas the higher cost of raw materials, however, is partly offset by the fact that although barium chemicals are not particularly bulky, differences in freight rates tend to equalize the final costs of the domestic and foreign products.

GROUND BARITE

Preparation

The preparation of ground barite varies in different plants and according to the condition of the ore. The general practice, however, is to crush the ore to about 1 inch and by log washing to remove the clay and earthy material. Often this washing is done at the mine and need not be repeated at the grinding mill. The next step is to grind the cleaned material, usually in tube mills; then the ground barite, suspended in water, is either screened or water floated in order to remove any coarse particles. If off-color, the finely ground barite is bleached with sulphuric acid, which removes the stains due to oxides of iron. After being washed several times with water, the bleached product is dried, pulverized, and packed in bags or barrels. Very little labor is required in the process of grinding after the ore is placed in the first or primary crusher until it reaches the packing room. The barite is conveyed mechanically or pumped, suspended in water, through the different processes.

Uses

The principal use of ground barite is as a filler. Since it is extremely heavy, chemically inert and white, it is admirably fitted for this purpose. The paint, paper, rubber, linoleum, and cloth industries all use considerable quantities to add weight and body to their products. By far the larger part - estimated at 50 to 70 per cent - of all the ground barite produced and sold in the United States is consumed in the paper and rubber industries.

As a rubber filler, bleached or unbleached, ground barite is employed where a soft rubber article is desired, as in rubber bands, tubing and the like, but not where strength and resistance to abrasion is required, as in auto tires. The bleached form is used in papers where weight is desirable with a high finish, as in Bristol board, playing cards, and plate paper. Natural barite is not often used in high-grade paints, but a limited quantity is employed in the cheaper grades and a small amount of the bleached mineral is used directly as an extender and loader and as a base for lake colors.

Finely ground barite mixed with a colloidal clay, such as bentonite, and a suitable suspending medium is effective in making up heavy liquids used as a seal in drilling for oil and gases. The heavy liquid holds back the gas flow until the casing can be placed. Finely divided iron oxide in suspension is employed for the same purpose.

A limited amount of unbleached ground barite is used in molded goods, such as buttons and hard rubber articles. One of its newer uses in the molded hard-rubber industry is as a component part of the protecting hard-rubber shield around X-ray tubes. Because of its high density, ground barite tends to absorb the rays and thereby protect the operator, except where the shield is open. Plaster containing a large admixture of ground barite is used on walls of X-ray rooms in order to limit the radiation of the rays. It is also used in making brick for the construction of X-ray laboratories, replacing in both cases the lining of sheet lead used to prevent the rays from escaping and doing possible injury.

Barium platino-cyanide has the property of becoming fluorescent under the action of X-rays and ultra-violet light rays. This barium compound is used in X-ray photography by coating the projecting screen with it. The shadows made by the rays are thus outlined on the screen.

A small quantity of ground barite is used in the manufacture of artificial ivory. Some barite is used as a flux in brass smelting, and is also used in the extraction of iron in reclaiming brass from junk.

Production

The production of ground barite has increased during recent years, but not as rapidly as the production of crude barite. During the years 1915 to 1920 from 31 to 50 per cent of the crude ore consumed in all industries was used for the manufacture of ground barite, but during the years 1921 to 1928 the proportion was only 20 to 29 per cent of the total. This decrease in the percentage of crude barite used for grinding is not due to any great extent to an increase in the imports of ground barite, but to a tremendous growth that that has taken place in the lithopone industry since 1922.

Table 1 and figure 1 show the amount of crude barite used in the manufacture of ground barite, and the percentage of the total for the years 1915 to 1928:

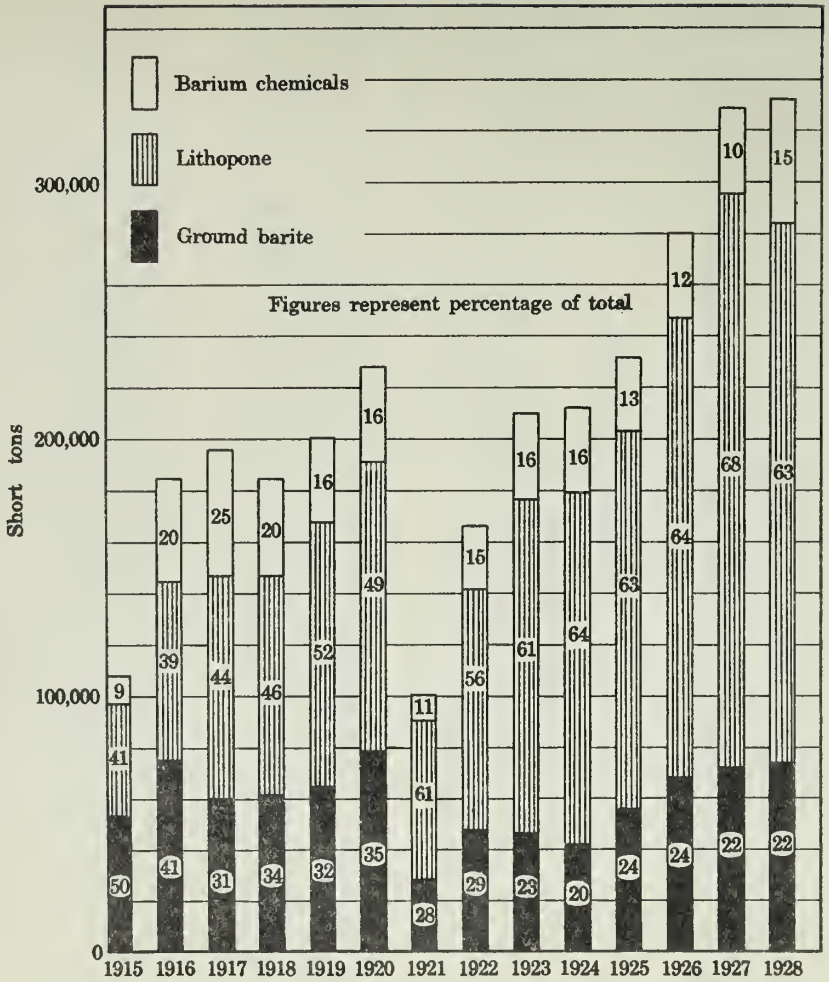


Figure 1. – Crude barite used in the manufacture of barium products

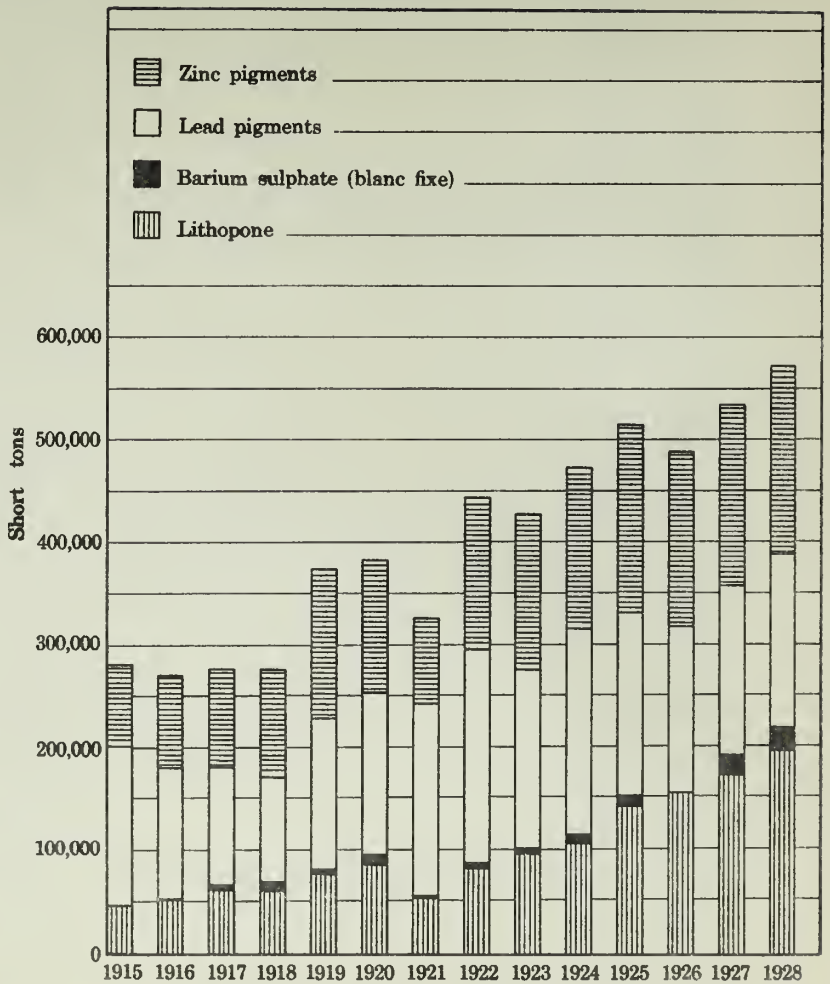


Figure 2. – Sales of barium, lead, and zinc pigments

Table 1. - Crude barites, both domestic and imported, used in the manufacture of barium products, 1915-1928 (in short tons)

Year	Used for grinding	Total used	Per cent of total
1915	53,903	108,622	50
1916	75,507	185,688	41
1917	60,132	196,039	31
1918	62,440	185,763	34
1919	64,922	201,586	32
1920	79,052	229,443	35
1921	28,296	100,607	28
1922	48,280	167,842	29
1923	47,713	210,587	23
1924	42,855	213,163	20
1925	56,851	232,685	24
1926	69,048	281,056	24
1927	73,119	330,190	22
1928	74,814	334,695	22

The grinding of barite is carried on chiefly in Missouri, which in recent years has produced 80 to 85 per cent of the barite ground in the United States from domestic ore. The other States which produce ground barite are Georgia, 5 to 6 per cent of the total; California, 4 to 6 per cent; and South Carolina, 6 to 8 per cent.

The dominant position held by Missouri in the ground barite industry is due to the grade and texture of the ore which is soft and easy to grind as compared with the hard crystalline variety found in other sections of the country.

Market and Prices

The market for ground barite is divided about equally between the Mid-Western field (St. Louis to Chicago) and the Atlantic Coast States. About 50 per cent of the ground barite used in the eastern market is supplied from Missouri, and the remainder is obtained in part from the Southern field and in part from imports (chiefly from Germany).

The principal rubber manufacturers who use ground barite as a filler in their products are located in and around Akron, Ohio, and in the vicinity of New York City. Most of the ground barite used by these companies is from Missouri, as the Southern ore enters into the rubber industry in only small quantities.

The paint industry, on the other hand, although it is located in practically the same two districts as the rubber industry, also includes Michigan, Illinois, and Indiana in the mid-western market, and Pennsylvania and Delavare in the eastern market. Though Missouri furnishes the paint industry with the larger part of its barite requirements, a greater proportion of Southern ore is used in this industry than in the rubber industry.

The price of ground barite has remained almost stationary in recent years; the average annual price as reported by producers to the Bureau of Mines ranges between $20 and $22 per short ton. Prior to the war the price ranged from $11.19 per ton in 1908 to $14.18 in 1913, and when hostilities began the price dropped to $12.95 in 1914 and then to $12.32 in 1915, but immediately advanced until in 1918 it reached $18.91 per ton.

Table 2. - Ground barite sold by producers in the United States,[1] 1908-1928

Year	Quantity short tons	Value	Average price
1908 [2]	28,415	$ 318,096	$11.19
1909 [2]	34,673	455,506	13.14
1910 [2]	38,165	475,001	12.45
1911 [2]	39,611	503,867	12.72
1912 [2]	38,225	495,895	12.97
1913 [2]	37,033	525,300	14.18
1914 [2]	35,815	463,858	12.95
1915 [2]	51,557	635,318	12.32
1916	65,440	964,802	14.74
1917	52,694	951,220	18.05
1918	55,086	1,041,608	18.91
1919	[3] 57,985	1,163,437	20.06
1920	65,748	1,381,868	21.02
1921	27,661	622,871	22.52
1922	46.176	933,722	20.22
1923	39,605	895,095	22.60
1924	38,296	784,881	20.50
1925	49.674	1,040,461	20.95
1926	57,812	1,186,563	20.52
1927	57,658	1,166,294	20.23
1928	64,425	1,332,228	20.68

1/ From domestic and imported ore.
2/ Sales of refined barite (domestic only).
3/ Figures cover production, not sales (1919).

Table 3 shows the prices of ground barite per long ton, washed and water floated, f. o. b. plants, bags extra, as quoted from the Engineering and Mining Journal.

Table 3. - Prices of Ground Barite (Washed and Water floated, f.o.b. mines, bags extra) for Georgia and Missouri, by year, each month.

	Georgia		Missouri					
	1928	1929	1924	1925	1926	1927	1928	1929
Jan.		$19.00	$26 $28	$23 $24	$23	$23	$23	$23
Feb.		$19.00	$23 $24	$23 $24	$23	$23	$23	$23
Mch.	$19.00	$19.00	$23 $25	$23 $24	$23	$23	$23	$23
Apr.	$19.00	$19.00	$23 $25	$23 $24	$23	$23	$23	$23
May	$19.00	$19.00	$23 $24	$23 $24	$23	$23	$23	$23
June	$19.00	$19.00	$23 $24	$23 $24	$23	$23	$23	$23
July	$19.00	$19.00	$23 $24	$23 $24	$23	$23	$23	$23
Aug.	$19.00	$18.00	$23 $24	$23 $24	$23	$23	$23	$23
Sept.	$19.00	$18.00	$23 $24	$23 $25	$23	$23	$23	$23
Oct.	$19.00		$23 $24	$23 $24	$23	$23	$23	
Nov.	$19.00		$23 $24	$23 $24	$23	$23	$23	
Dec.	$19.00		$23 $24	$23 $24	$23	$23	$23	
Average:	$19.00		$23.88	$23.58	$23	$23	$23	
High	$19.00	$19.00	$28.00	$24.00	$23.00	$23.00	$23.00	
Low	$19.00	$19.00	$23.00	$23.00	$23.00	$23.00	$23.00	

IMPORTS AND EXPORTS

The following Table No. 4 shows the imports of ground barite for consumption for the years 1910 to 1928, inclusive.

The imports of ground barite, though formerly important, now play only a small part in the ground barite industry of the United States. Since 1923 they have never exceeded 3,000 tons in any one year. The declared value per ton, as reported in the statistics, plus freight, duty and other charges, indicates that the apparent delivered cost is as high, or higher, than the price obtained for the domestic product.

As previously stated, about 50 per cent of the production of ground barite is consumed along the Atlantic seaboard where it comes in direct competition with German ground barite, but due to the careful preparation of the domestic product as regards fineness, uniformity of quality, etc., and the fact that shipments are made promptly from stocks at American mills, Germany has failed to obtain as strong a foothold in the ground barite industry as it has in the case of crude barite.

There are no recorded exports of ground barite, though it is understood that small quantities are shipped to paint, paper and rubber plants in Canada.

Table 4. - Imports of ground barite for consumption in the United States, 1910-1928

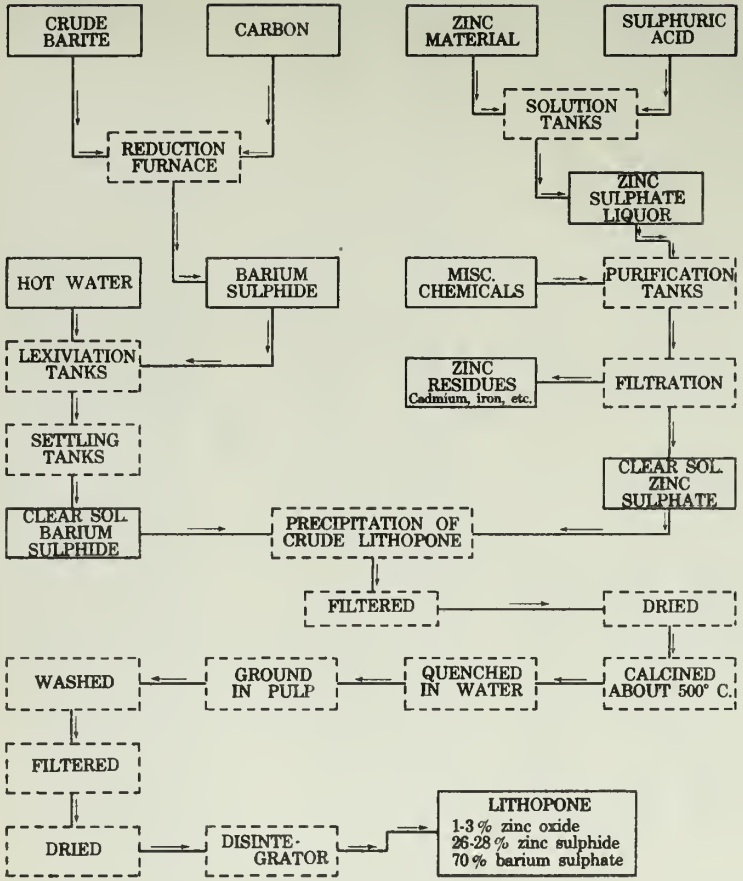
Year	Rate of duty	Quantity short tons	Value	Price per ton
1910	$5.25 per ton	3,023	$ 27,191	$ 8.99
1911	do	3,655	26,462	7.24
1912	do	3,012	21,642	7.19
1913	do	5,145	36,819	7.16
1914 1/	do	1,107	7,520	6.79
1914 2/	20 per cent ad va-	4,919	35,105	7.14
1915	do lorem	1,966	14,997	7.63
1916	do	-	2	-
1917	do	235	3,813	16.23
1918	do	-	-	-
1919	do	-	-	-
1920	do	245	3,017	12.31
1921	do	895	16,942	18.93
1922 3/	do	3,027	43,392	14.34
1922 4/	$7.50 per ton	1,166	16,061	13.77
1923	do	5,377	88,736	16.50
1924	do	2,527	38,292	15.15
1925	do	2,860	40,008	13.99
1926	do	2,988	44.781	14.99
1927	do	2,910	38,568	13.25
1928	do	2,766	33,378	12.07

1/ From July 1 to Oct. 3, 1913.
2/ From Oct. 4, 1913 to June 30, 1914.
3/ From Jan. 1, 1922, to Sept. 21, 1922.
4/ From Sept. 22 to Dec. 31, 1922.

LITHOPONE

Preparation

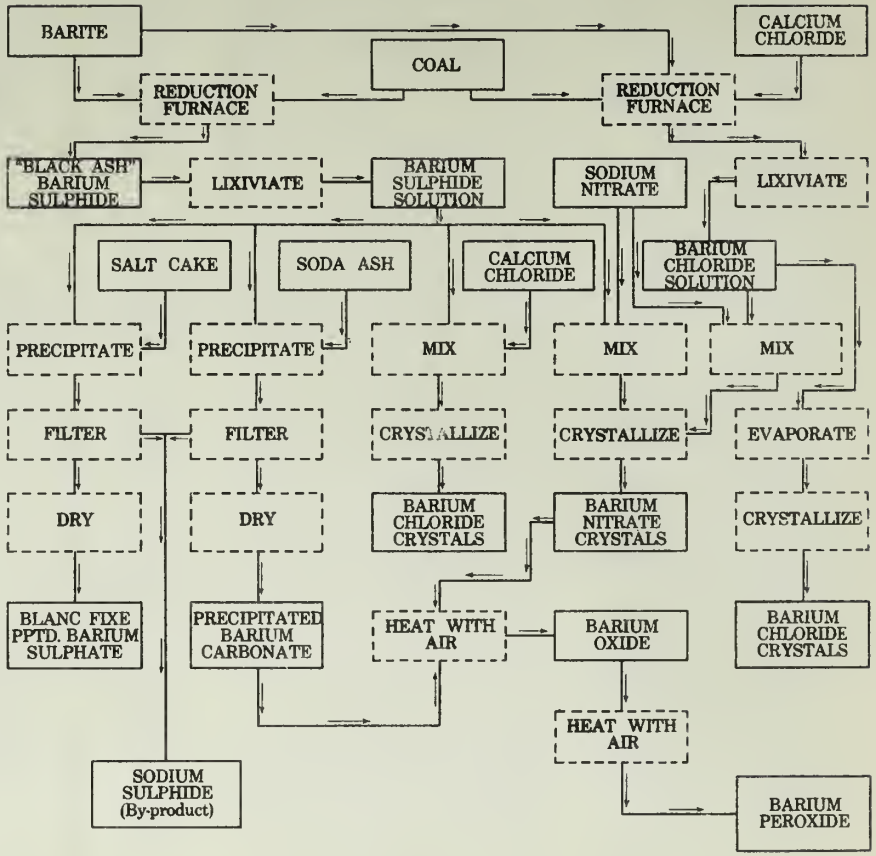
The basic raw materials necessary for the manufacture of lithopone are crude barite; coal, coke, or petroleum breeze; zinc; and sulphuric acid. Other chemicals, such as ferric oxide, alumina, lime, and magnesia are required in the process of manufacture, but in minor quantities. Large quantities of clean, soft water are required in making up solutions and in the washing of lithopone to remove soluble impurities. The manufacture of lithopone requires the preliminary production of a solution of barium sulphide and a solution of zinc sulphate. For the preparation of barium sulphide, the crude barite is first crushed and mixed with the proper proportions of pulverized coal or other forms of carbon. This proportion varies slightly with individual manufacturers; it is generally one part of coal to four parts of barite by weight. After heating the mixture for about four hours in a rotating furnace, the barium sulphate is reduced to barium sulphide. The crude sulphide (black ash), which contains about 70 per cent barium sulphide, is then treated with hot water, and the resulting solution of barium sulphide is clarified.



LEGEND

Solid lines.----Raw, intermediate, or finished products
Broken lines.--Apparatus, or steps in process

Figure 3. – Flow sheet for the manufacture of lithopone



LEGEND

Solid lines.----Raw, intermediate, or finished products
Broken lines.--Apparatus, or steps in process

Figure 4. – Flow sheet of the barium chemical industry

The zinc sulphate solution is prepared by dissolving zinc--in the form of zinc ash, scrap, skimmings, slab zinc, or roasted ore--in dilute sulphuric acid. Since the zinc materials usually contain as impurities other metals (iron, copper or cadmium) which would form colored products on treatment with barium sulphide, the zinc sulphate solution must be purified. Various chemicals are used for this purpose, and this purification, since it represents one of the most important steps in the entire process of manufacture, requires careful chemical control.

Crude lithopone is precipitated from the purified zinc sulphate solution as a white insoluble substance by adding a hot solution of barium sulphide. The crude lithopone is filtered off, dried, and then calcined at about 500° C. in a muffle furnace, which may or may not utilize waste heat from the barite reduction furnace. The hot lithopone is thrown into water, ground, washed several times with water to remove soluble impurities, filtered, and dried. After being run through a disintegrator and reduced to a fine powder, the dry product is packed in bags or barrels for shipment. In some cases the lithopone is air-floated to insure a uniform product. The process is shown graphically in Figure 3.

The value of lithopone depends upon the amount of zinc sulphide in it as well as upon the fineness and whiteness of the product.

Following are several analyses of typical samples of lithopone:

Content	German	English[4/]	
	1	2	3
$BaSO_4$	64.36	67.59	72.80
ZnS	30.98	27.82	24.14
ZnO	0.55	1.82	0.88
Not reported	4.11	2.77	2.18
	100.00	100.00	100.00

Inferior qualities sometimes contain natural barite which reduces the opacity and covering power and causes them to give a yellowish-white tint when used in linoleum. The natural mineral may be detected by means of the microscope, the precipitated barium sulphate appearing as a fine powder composed of minute crystals of uniform size, whereas the natural barite consists of transparent irregular fragments of larger size.

Uses

Lithopone is used extensively as a white pigment in flat and enamel wall paints for interior use. It is also used in large quantities as a filler in rubber goods, paper, linoleum, oilcloth, and in window-shade cloth. Lithopone has not been used extensively in outside paints because it darkens when

4/ Jour. Soc. Chem. Ind., Sept. 15, 1902, p. 1145
Kochs, E., and Seyfert, F., Pigments and Paints.

exposed to sunlight or excessive moisture. It usually becomes white again, however, upon drying out, and several brands of lithopone paint have been placed on the market in recent years which are said to withstand exposure to sunlight and moisture.

Production and Sales

The rapid growth during recent years in the output of crude barite was the direct result of a remarkable and rapidly increasing demand for lithopone as a paint pigment.

Table 5 shows the amount of crude barite used in the manufacture of lithopone and the ratio of this consumption to the total barite used in all industries for the years 1915 to 1928, inclusive.

Up to and including the war period the amount of crude barite used for the manufacture of lithopone never exceeded 50,000 short tons annually, while the percentage consumed in this industry never exceeded 46 per cent. Since the war, however, the tonnage of crude barite sold for this use increased rapidly until in 1928, 211,592 short tons were consumed. The percentage of the total consumed in all industries likewise advanced rapidly, until in 1927 68 per cent of the total sales of crude barite were used in the manufacture of lithopone (63 per cent in 1928).

Table 5. - Crude barite, both domestic and imported, used in the manufacture of barium products, 1915-1928 (in short tons)

Year	Used in lithopone	Total used	Per cent of total
1915	44,503	108,622	41
1916	71,898	185,688	39
1917	86,065	196,039	44
1918	85,282	185,763	46
1919	103,688	201,586	52
1920	113,181	229,443	49
1921	61,359	100,607	61
1922	94,259	167,842	56
1923	128,547	210,587	61
1924	137,127	213,163	64
1925	146,693	232,685	63
1926	178,889	281,056	64
1927	222,791	330,190	68
1928	211,592	334,695	63

The marketed production of lithopone has followed closely the sales of crude barite to the lithopone manufacturers, the sales totalling 8,000 to 10,000 tons less annually than that reported as consumed for its manufacture. As may be seen in Table 6, the marketed production prior to the war never exceeded 30,000 short tons, the average for the years 1910 to 1913 being 20,856 short tons. During the war period (1914 to 1918) the production increased rapidly, due to the growing demand and the discontinuance of lithopone imports.

The average for this five-year period was 51,344 tons, an increase of 146 per cent over the pre-war period. Following the war, the marketed production fluctuated considerably until after the general depression year 1921, when production returned to its normal growth. The average production for this period (1919 to 1928) increased to 119,619 short tons, an increase of 133 per cent over the pre-war period.

The price per ton obtained for lithopone on the American market advanced rapidly with the advent of war until in 1920, the all-time high price of $139.69 per ton was reached. Since then, however, the price has again fallen to a normal increase over that of the pre-war period, the price in 1928 reaching $95.15 per ton.

Table 6. - Marketed production of lithopone, 1910-1928, in short tons

Year	Quantity	Value	Price per ton
1910	12,655	$ 916,512	$ 72.42
1911	16,866	1,243,108	73.70
1912	24,220	1,702,119	70.28
1913	29,685	2,170,445	73.12
1914	32,819	2,490,530	75.89
1915	46,494	3,760,472	80.88
1916	51,291	5,798,927	113.06
1917	63,713	7,406,896	116.25
1918	62,403	7,923,276	126.97
1919	78,365	10,218,850	130.40
1920	89,373	12,484,925	139.69
1921	55,016	6,281,563	121.45
1922	83,360	9,214,040	110.53
1923	98,199	11,608,443	118.21
1924	109,469	12,531,397	114.47
1925	145,019	15,186,147	104.72
1926	159,931	16,062,197	100.43
1927	176,994	17,163,620	96.97
1928	200,468	19,073,914	95.15

Market and Prices

The demand for lithopone, though well distributed throughout the United States, is concentrated chiefly along the Atlantic seaboard, in the Middle-Western States (from St. Louis to Chicago), and in the Pacific Coast States, principally California and Washington.

Prior to the war the lithopone requirements of the United States were furnished almost solely by eastern plants using both domestic and imported crude barite. The imported ore, however, was delivered to consumers at a price much lower than that of the domestic material. Before the war Missouri was the principal source for crude ore for lithopone requirements in the East, which necessitated a long freight haul to the lithopone plants on the Atlantic seaboard.

With the advent of war, crude barite imports ceased, and the eastern manufacturers of lithopone, in order to secure a source of raw material, developed the barite deposits in the Southern States, principally those in Georgia, Tennessee, and Kentucky.

Later, in order to meet the growing demand for lithopone in the Mid-Western States, eastern paint manufacturers constructed plants at St. Louis, Mo., and in neighboring States, utilizing locally produced crude barite. This new industry expanded rapidly and with the termination of the war was able to hold its own in seeking an outlet for its finished product.

About 1926 the lithopone industry was started in California using locally produced barite, and as the industry did not meet with competition from either the Missouri or eastern products it prospered.

Table 14 shows the prices obtaining for lithopone for the years 1922 to 1928, as quoted by the Oil, Paint and Drug Reporter. A slight decrease is noticeable during this period. In 1922 the price ranged from 6 to 6¼ cents per pound, f. o. b. New York, and represented a slight decline from the prices quoted in the preceding year. In 1923 and again in 1924, there were marked advances, the price ranging from 6 to 7¼ cents per pound during the former year, and from 6-3/8 to 8-5/8 cents during the latter year. In 1925, however, quotations dropped to 6-1/8 to 6-5/8 cents, and in 1928 the prices ranged from 5-3/4 to 5-7/8 cents, the lowest price for this product over a range of 10 years.

Imports and Exports

The imports of lithopone into the United States have never exceeded 11,000 short tons a year.

Table 7 shows the tonnage, the declared value, and the average price per ton for the years 1913 to 1928, inclusive. Prior to 1913, the imports of lithopone were included in the figures for "zinc sulphide white."

With the entrance of the United States into the World War the imports of lithopone ceased entirely, the material having come principally from Germany. In 1919, when business relations were again restored, the United States imported only 739 tons, valued at $122,708, or the high price of $166.05 per ton; but in 1922 the imports had grown to 10,763 tons, valued at $885,671, or $82.29 per ton, representing less than half the price per ton of 1919. In 1923 the imports were 10,440 tons, valued, however, at only $776,383, or $74.37 per ton, and in 1924 they dropped to 6,830 tons, valued at $543,025, or $79.51 a ton. In 1928 they increased again to 9,885 tons, valued at $808,200, or $81.76 per ton.

Table 7. - Lithopone imported for consumption into the United States, 1913-1928

Year	Quantity short tons	Value	Price per ton
1913	2,363	$143,474	$61.99
1914	3,990	271,310	68,00
1915	2,044	137,816	67.42
1916	2,341	405,730	173.31
1917	224	29,199	130,35
1918	-	-	-
1919	739	122,708	166,05
1920	1,714	263,240	153.58
1921	5,247	432,019	82.34
1922	10,763	885,671	82.29
1923	10,440	776,383	74.37
1924	6,830	543,025	79,51
1925	6,330	478,395	75.58
1926	8,686	661,700	76.18
1927	7,979	656,166	82.24
1928	9,885	808,200	81.76

Prior to 1922 the exports of lithopone from the United States were not separately recorded. Since that year, however, there has been a steady growth in our export trade in this commodity, as is shown in Table 8.

The declared export value per ton has been slightly above that of the domestic sales value and has followed closely the price trend of domestic sales.

Table 8. - Lithopone exported from the United States, 1922-1928. 6/

Year	Short tons	Value	
		Total	Average
1922	1,616	$169,982	$105.19
1923	1,485	176,624	118.94
1924	923	104,783	113.52
1925	1,287	132,771	103.16
1926	1,941	200,412	103.25
1927	2,110	222,585	105.49
1928	3,326	337,523	101.48

6/ Not separately recorded prior to 1922.

Correlation of Sales of Barium Pigments With Lead and Zinc Pigments

Barium pigments, especially lithopone, have certain advantages over both white lead and zinc oxide. Lithopone itself is whiter than white lead and equal in whiteness to the best grade of French-process zinc oxide. Unlike

white lead it is not affected by sulphur or hydrogen sulphide and is nonpoisonous, whereas lead paint is so poisonous that certain countries have prohibited its use for interior decoration. Lithopone does not blacken by combining with lead contained in pigments or driers with which it is mixed. In opacity or drying power it is superior to white lead, and about equal to zinc oxide, but this claim can not be sustained with respect to lithopone when employed in a mixed paint, as the larger quantity of oil used with lithopone reduces the covering power of the paint.

As an exterior paint lithopone is said not to give complete satisfaction when used alone. In direct sunlight it is likely to discolor and turn gray, and its surface becomes chalky and checked. However, the claim is made that by mixing small quantities of other white pigments, such as zinc oxide and whiting (calcium carbonate or chalk) with lithopone, these objections are removed. Such pigments, made in part of lithopone, are on the market and are said to be eminently satisfactory for exterior use. A mixture of 60 per cent of lithopone, 20 per cent of zinc oxide, and 20 per cent of whiting ground in pure linseed oil makes a satisfactory paint for exterior use. Therefore lithopone comes into direct competition with lead and zinc pigments in exterior paint.

Table 9 and figure 2 correlate the sales of barium pigments with the sales of lead and zinc pigments for the years 1910 to 1928.

Table 9. - Barium, lead, and zinc pigments sold by producers in the United States 1910-1928 (in short tons)

Year	Lithopone	Barium sulphate (blanc fixe)	Lead pigments 1/	Zinc pigments 2/
1910	3/ 12,655	(4)	143,810	5/ 50,592
1911	3/ 16,866	(4)	132,612	5/ 46,122
1912	3/ 24,220	(4)	146,833	5/ 64,032
1913	3/ 23,685	(4)	142,626	5/ 70,622
1914	3/ 32,819	(4)	159,474	5/ 70,619
1915	46,494	(4)	156,101	5/ 79,572
1916	51,291	1,229	128,979	5/ 90,276
1917	63,713	3,537	115,200	5/ 97,573
1918	62,403	9,522	102,888	5/ 102,778
1919	78,365	5,227	148,158	145,230
1920	89,373	8,046	158,107	129,904
1921	55,016	2,471	181,851	90,432
1922	83,360	7,349	208,756	148,078
1923	98,199	6,761	174,822	150,491
1924	109,469	7,796	202,066	158,199
1925	145,019	11,748	178,901	183,104
1926	159,931	(6)	162,084	170,782
1927	176,994	13,395	171,177	177,310
1928	200,468	19,081	169,947	185,129

1/ White lead (dry and in oil) and white lead sulphate.
2/ Zinc oxide and leaded zinc oxide.
3/ Marketed production.
4/ Not available.
5/ Zinc content of zinc pigment- domestic and imports.
6/ Bureau of Mines not at liberty to publish figures.

While the sales of lead pigments have been quite variable over this period, those of zinc and especially of lithopone have shown favorable increases.

The sales of lithopone during the pre-war period (1910 to 1913) averaged 20,856 short tons annually, while during the war period (1914 to 1918) they advanced to an average of 51,344 tons, an increase of 146 per cent. The post-war period sales averaged 119,619 tons annually, an increase of 133 per cent over the war period, and of 474 per cent over the pre-war period.

The sales of lead pigments, while larger, did not show such a tremendous growth. The average sale of lead pigments for the pre-war period (1910 to 1913) was 141,470 tons, or about 7 times that of lithopone for the same period. During the war, however, it decreased, the average being only 132,528 tons, or 6 per cent less than during the previous period. Since the war there has been only a relatively slight increase, the average sale for the years 1919 to 1928 being 175,587 tons, or about 24 per cent more than during the pre-war period.

The sales of zinc pigments, however, have been more favorable. During the pre-war period they averaged 57,842 tons, nearly three times that of lithopone and about one-third that of lead pigments. For the war period they increased to an average of 88,163 tons, or 34 per cent over the previous period, while from 1919 to 1928, inclusive, they averaged 153,866 tons, representing increases of 43 per cent over the war period and of 62 per cent over the pre-war period.

The tremendous growth in the lithopone industry, as compared with its two closest competitors, lead and zinc pigments, augurs well for the future of the industry.

BARIUM CHEMICALS

Preparation

Barium chemicals have important industrial uses in the United States. The most important of these chemicals are as follows: Blanc fixe, or precipitated barium sulphate (which, although practically identical with ground barite in chemical composition, has different physical properties), barium carbonate, chloride, nitrate, hydroxide, and dioxide.

Barium sulphide, which is prepared by heating a mixture of crude barite and some form of carbon (see p. 8), is used as the starting point for the preparation of barium chemicals.

Both barite and witherite are used in the United States as raw material for the manufacture of barium chemicals, but the quantity of the latter consumed for this purpose is relatively small and is largely imported from England. The manufacturers of barium chemicals prefer to use the washed, high-grade barite of the soft type, but they can and do use barite which could not be used in the preparation of the highest grades of ground barite.

Barium carbonate is precipitated from a solution of barium sulphide by sodium carbonate (soda ash), and sodium sulphide is obtained as a by-product. Barium carbonate can also be prepared by passing a current of carbon dioxide gas through the solution of barium sulphide.

The method of producing blanc fixe or precipitated barium sulphate depends on the physical properties which are desired for the finished product. It was formerly made only by treating a solution of barium chloride with sodium sulphate, or sulphuric acid. It is now usually made by precipitation from a solution of barium sulphide by means of salt cake (sodium sulphate). This method produces a much denser product than that produced from barium chloride and in addition yields sodium sulphide as a valuable by-product. It is also made as a by-product of the manufacture of hydrogen peroxide when barium peroxide is treated with sulphuric acid. For certain color purposes blanc fixe is precipitated from hot, concentrated solutions, thus giving a crystalline product which is a valuable base for the manufacture of very brilliant colors.

Barium chloride may be produced by treating witherite (natural barium carbonate) with hydrochloric acid. It is usually produced in this country directly from crude barite by roasting with coal and calcium chloride. It may also be prepared by treating a solution of barium sulphide with calcium chloride.

Barium nitrate is produced by adding sodium nitrate (Chile saltpeter) either to a solution of barium chloride or to a solution of barium sulphide. It is also produced by dissolving witherite in a solution of nitric acid. The barium nitrate thus produced can be converted into barium oxide, dioxide, or hydroxide.

Barium hydroxide, or caustic barite, is prepared on a large scale by heating barium sulphide in earthenware retorts into which a current of moist carbonic acid is passed, after which superheated steam is passed over the resulting heated carbonate.

Barium peroxide, or dioxide, is prepared from either barium nitrate or barium carbonate. The barium nitrate, or carbonate, is first converted into barium oxide by heating in a closed apparatus, and the barium oxide is then converted into barium peroxide by heating at about 700° C. in a current of air free from carbon dioxide and moisture.

Whenever a sodium salt is added to a solution of barium sulphide in the preparation of barium salts, sodium sulphide is produced as a valuable by-product in addition to the desired barium product. This is an important product of the barium chemical industry, and has a large consumption by the American dye industry in the preparation of sulphur colors, especially sulphur black.

Figure 4 shows the steps, apparatus, and raw materials, used in the manufacture of barium chemicals.

Uses

Blanc fixe, or permanent white, the precipitated barium sulphate, is the best grade of barium sulphate for pigment purposes, both as to whiteness and fineness of product. It is largely used where a pure white pigment or filler is desired, as in paints, rubber goods, linoleum, oilcloth, and glazed paper. It is also used in lithographic inks and as a base for lake colors. The battleship gray used extensively by the United States Navy contains about 45 per cent of blanc fixe. Under the name of chemically pure barium sulphate it is used as an indicator in X-ray photography.

Blanc fixe is placed on the market in pulp and as a dry powder. The pulp is a paste containing about 30 per cent of water, and is used chiefly by the paper and lithographic ink manufacturers. The dry powder is used for other pigment purposes.

Barium carbonate is a white insoluble salt which finds its largest use in the ceramics industry, where it is employed to prevent scumming on the surface of clay products and for the manufacture of optical glass, for which purpose a chemically pure product is required.

In the past various compounds of barium have been used to prevent the formation of scum, but at present they have practically all been replaced by barium carbonate. Nearly all of the soluble salts which cause this scum are sulphates, and calcium sulphate is by far the most troublesome in this respect. If a sufficient quantity of barium carbonate be added to a clay containing calcium sulphate, the calcium sulphate, which is a relatively soluble salt, will be broken up, but the products of the reaction, - barium sulphate and calcium carbonate, - being relatively insoluble, will not form a scum.

Barium carbonate is also used in growing quantities in the manufacture of flat wall paints. These paints, when containing about 45 per cent of precipitated barium carbonate, produce a velvety finish. Witherite (natural barium carbonate) can not be used for this purpose and should not be confused with precipitated barium carbonate. Barium carbonate is also used as a raw material for making barium peroxide and in the manufacture of certain kinds of enameled ironware.

Barium chloride is a white barium salt which is fairly soluble in water. It crystallizes with two molecules of water and is usually sold in the crystalline or powdered form. It is largely used in the manufacture of blanc fixe, lake colors, and as a mordant for fixing acid dyes on an inert base such as aluminum hydroxide or blanc fixe. Barium chloride is also used in the purification of salt, as a water softener, as a chemical reagent in sulphur determinations, to some extent in the ceramics industry, and in the manufacture of certain photographic chemicals. It has certain medicinal uses.

Barium nitrate is a soluble barium salt used principally for the manufacture of barium peroxide or dioxide. It is also used in pyrotechnics for making green fire and green signal lights and in the manufacture of certain explosives. It also has certain medicinal uses similar to those of barium chloride.

Barium hydroxide, or caustic barite, has little commercial use in the United States. During periods of high prices for caustic potash, barium hydroxide is used to convert sulphate of potash into caustic potash. It has, until recently, been used in the refining of beet sugar, but as strontium or lime act in similar manner and are not poisonous, they have been substituted, especially the latter, for barium hydroxide. A small quantity, however, is still used in a western sugar refinery for this purpose.

Barium peroxide, or dioxide, finds its only use in this country in the manufacture of peroxide of hydrogen.

Barium monoxide is used for the preparation of barium peroxide, in the manufacture of certain varieties of optical glass, and in Europe in refining beet sugar.

Barium sulphide is the intermediate compound from which the majority of the other barium chemicals are made, and is one of the necessary materials in the manufacture of lithopone. It is also used with lime as a depilatory.

Barium chromate, lemon yellow, or yellow ultramarine, is used as a yellow pigment.

Barium chlorate is used in pyrotechnics and in dyeing.

Barium manganate has been used to some extent as a green pigment.

A mixture of barium sulphate, sulphide and carbonate has been patented by E. E. Fisher (United States Patents 1,665,693 and 1,665,694) for use in mixes for the manufacture of pressed or blown glass ware. This mixture is claimed to be particularly beneficial where sharp corners and intricate patterns have to be completely filled. The barium chemicals used in glass making must be virtually free from iron, silica and dolomite.

Production and Sales

Prior to the World War the United States was almost wholly dependent on imports for its supply of barium chemicals. There was practically no domestic production of the barium salts such as barium dioxide, chloride, nitrate, and blanc fixe. At that time the only chemical product manufactured in this country on a large scale which utilized barite as a raw material was lithopone. The shutting off of imports with the outbreak of the war gave a great stimulus to the establishment of a barium chemical industry in the United States. In addition there were large war demands for certain barium salts (barium peroxide, blanc fixe, and barium nitrate), which greatly increased the domestic production and tended to develop an industry in the United States.

According to Table 10 and figure 1 the use of crude barite for the manufacture of barium chemicals has not shared the increases exhibited in the ground barite or lithopone industries. At no time has the percentage of crude barite used in the chemical industry exceeded 25 per cent of the total used in all industries, and generally it has been much less.

During the period from 1915 to 1920, inclusive, the barium chemical industry consumed on an average of 24,442 short tons annually of both domestic and imported barite, 18 per cent of that used in all industries.

The average annual consumption of crude barite, both domestic and imported, in the chemical industry during the period from 1921 to 1928, inclusive, was 31,074 tons, an increase of 27 per cent over the preceding period (1915-1920), but only 13 per cent of the total used in all industries.

Table 10. - Crude barite, both domestic and imported, used in the manufacture of barium products, 1915-1928 (in short tons)

Year	Used in barium chemicals	Total used	Per cent of total
1915	10,216	108,622	9
1916	38,283	185,688	20
1917	49,842	196,039	25
1918	38,041	185,763	20
1919	32,976	201,586	16
1920	37,210	229,443	16
1921	10,952	100,607	11
1922	25,303	167,842	15
1923	34,327	210,587	16
1924	33,181	213,163	16
1925	29,141	232,685	13
1926	33,119	281,056	12
1927	34,280	330,190	10
1928	48,289	334,695	14

Tables 11 and 12 show the marketed production of the principal barium chemicals in the United States over a period of years. The total marketed production has registered a steady growth since the world-wide depression year of 1921 and during 1927 and 1928, has exceeded the sales of barium chemicals of any year prior to 1921. The total value has shown an upward trend, although the price per ton has decreased far below that obtaining during and preceding the war.

Table 11. - Marketed production of barium chemicals[1] in the United States, 1915-1928, in short tons

Year	Quantity	Value	Price per ton
1915	8,823	(2)	(2)
1916	16,792	$1,962,520	$116.87
1917	22,503	(2)	(2)
1918	23,186	(2)	(2)
1919	20,013	1,519,788	75.94
1920	20,760	1,743,634	83.99
1921	5,487	412,275	75.14
1922	13,901	1,562,216	112.38
1923	16,803	1,231,646	73.30
1924	18,154	1,213,900	66.87
1925	18,654	1,207,770	64.75
1926	21,911	1,325,262	60.48
1927	23,943	1,517,994	63.40
1928	33,969	2,045,156	60.21

1/ Barium chemicals include dioxide, carbonate, chloride, hydroxide, nitrate, sulphate (blanc fixe), and sulphide.
2/ Not available.

Of the barium chemicals sold in the United States, blanc fixe, or precipitated barium sulphate, is by far the most important. During recent years this commodity has accounted for about 33 per cent of the total quantity of the barium chemicals sold. Barium carbonate is the next important of the chemicals and accounts for about 25 per cent of the total.

Table 12. - Barium chemicals sold by producers in the United States, 1918-1928, in short tons

Year	Barium carbonate	Barium chloride	Barium sulphate (blanc fixe)	Other barium chemicals 1/	Total
1918	7,661	4,530	9,522	1,473	23,186
1919	2/ 7,135	4,509	5,227	3,142	20,013
1920	7,484	3,084	8,046	2,146	20,760
1921	1,956	(3)	2,471	1,060	5,487
1922	2,281	2,022	7,349	2,249	13,901
1923	6,362	3,301	6,761	379	16,803
1924	6,058	3,859	7,796	441	18,154
1925	4,962	(3)	11,748	1,944	18,654
1926	5,394	4,592	(3)	11,925	21,911
1927	5,969	3,708	13,395	871	23,943
1928	7,626	5,224	19,081	2,038	33,969

1/ Figures for "other barium chemicals" cover different chemicals for each year.
2/ Figures for 1919 cover production, not sales.
3/ Included under "other barium chemicals."

Markets and Prices

The barium chemical industry in the United States is well situated as regards markets. The demand for barium chemicals is more limited than that for lithopone or other barium products, and the plants are generally located near their market instead of at the source of raw material, though several are in fairly close promimity to domestic deposits.

The chief markets for barium chemicals are in the East, and the industry comes into direct competition with foreign products, which prior to the war satisfied fully two-thirds of the domestic requirements. The eastern plants are so situated that they are able to use either domestic or imported barite for their needs, and are thus able to furnish their market at the lowest possible price.

The barium chemical industry in the Mid-Western States use the locally produced barite and distribute the finished products to the market areas to the north and west of them.

The western producer of barium chemicals finds his market in the immediate vicinity, and has no competition from either Mid-Western or imported barium chemicals.

The prices of certain barium chemicals are shown in Table 14, for the years 1922 to 1928, inclusive.

Imports and Exports

Table 13 shows the imports for consumption of certain barium chemicals for the years 1902 to 1928, inclusive. There has been a general increase in practically all items, with witherite, blanc fixe, and barium nitrate showing the larger gains. The war curtailed our imports to a great extent, and it was not until 1920 that they returned to anything like the pre-war levels.

Prior to the war practically 90 per cent of the imports of barium chemicals into the United States were represented by three barium salts--namely, precipitated barium carbonate, barium chloride and barium dioxide. Within recent years, however, there has been a considerable growth in the imports of natural barium carbonate, blanc fixe, and barium hydroxide, until they represent nearly one-half of the total imports of barium chemicals.

Previously, these chemicals were generally imported from Germany, but now England, Belgium, and Italy furnish nearly one-half of our import trade in them. Our requirements of witherite, however, are furnished almost solely by England.

It is a question whether the near future will see the same steady growth in the importation of barium chemicals (excepting witherite) as has taken place during recent years. The industry in this country seems now to be on a firm economic and financial basis and able to take care of domestic

requirements, furnishing the consuming markets with products equally as good and at prices comparable with the imported products. Figure 5 shows the location of barite deposits and barium products plants in the United States.

Table 13. - Value of barium compounds imported into the United States, 1902-1928.

Year	Witherite, barium carbonate		Barium binoxide	Barium chloride	Blanc fixe	Barium nitrate	Barium hydroxide
1902	$12,777		$66,746	$46,905	$25,933	-	-
1903	25,762		84,549	68,762	35,466	-	-
1904	46,133		102,076	43,694	50,901	-	-
1905	45,073		111,856	47,386	53,112	-	-
1906	55,405		152,403	65,242	61,961	-	-
1907	24,552		167,519	79,333	85,713	-	-
1908	22,159		181,533	42,291	73,131	-	-
1909	31,584		255,013	47,352	65,427	-	-
1910	25,229		341,631	35,614	67,975	-	-
1911	27,351		270,917	28,896	71,049	-	-
(1)	Natural	"Precipitated barium carbonate"					
1912	$15,777	$ 9,938	252,320	27,655	70,327	-	-
1913	13,116	38,949	239,000	37,620	62,785	-	-
1914	8,084	28,221	332,709	68,866	32,619	-	-
1915	12,165	2,786	218,776	31,295	18,501	-	-
1916	18,169	-	6,590	608	17,819	-	-
1917	17,321	1,554	-	-	3,333	-	-
1918	14,134	-	-	-	-	-	-
1919	4,739	2,666	-	19,846	90	-	-
1920	61,284	37,462	64,447	151,778	8,485	-	-
1921	25,188	80,603	127,631	72,621	22,470	-	-
1922	39,114	210,250	189,334	112,785	76,445	$16,795 2/	$6,573 2/
1923	41,823	79,482	152,270	75,014	32,063	59,766	16,872
1924	56,415	128,217	120,374	100,810	50,465	54,638	8,241
1925	63,634	160,164	70,553	67,306	81,439	58,192	5,605
1926	80,679	189,167	8,157	49,426	136,697	49,494	9,789
1927	65,821	119,014	1,264	45,285	107,179	65,284	12,439
1928	82,434	116,241	1,310	25,253	168,923	69,010	10,031

1/ Figures prior to 1912 included both "natural" and "precipitated barium carbonate."

2/ Not separately recorded prior to the tariff of Sept. 22, 1922.

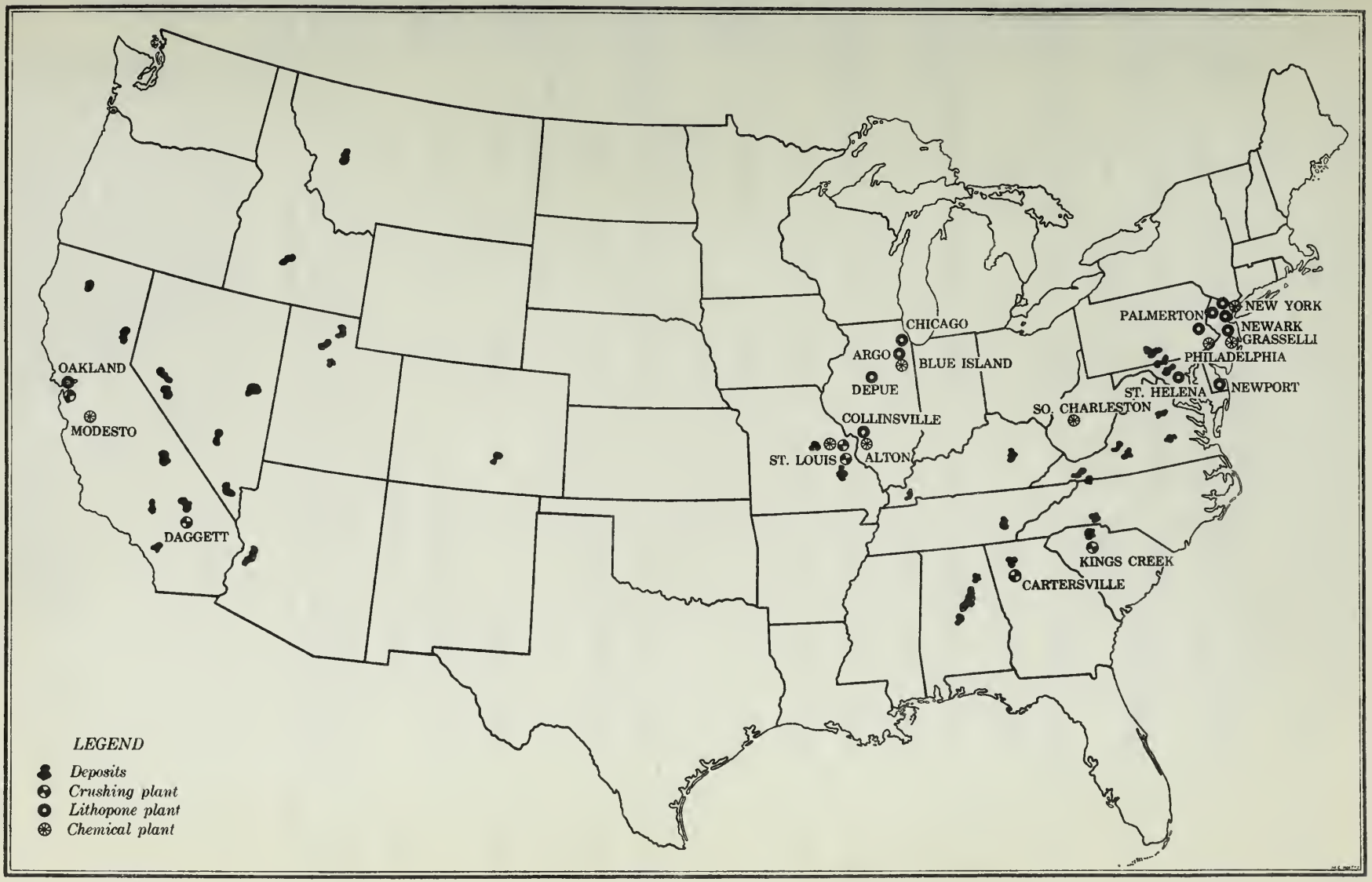


Figure 5. – Location of barite deposits and barium products plants



Table 14. - Prices of barite and barium products 1/

		1922	1923	1924	1925	1926	1927	1928
Barite, crude (short ton)	High	$ 7.25	$ 7.77	$ 9.00	$8.50	$8.50	$8.00	$7.50
	Low			8.00		8.00		6.50
Barite, ground (short ton)	High	26.00	28.00	26.00	23.00	23.00	23.00	23.00
	Low	23.00	25.00	22.00	22.000			
Blanc fixe (dry) (pound)	High	.04¼	.04¼	.05	.05	.04½	.04 3/8	.04½
	Low			.04	.04	.04	.04¼	.03 5/8
Blanc fixe(pulp) (short ton)	High	50.00	60.00	70.00	70.00	65.00	65.00	$60.00
	Low	40.00	40.00	50.00	55.00	53.00	60.00	40.00
Barium carbonate (short ton)	High	85.00	90.00	90.00	65.00	55.00	54.00	60.00
	Low	48.00	66.00	63.00	54.00	50.00	48.00	47.50
Barium chlorate (pound)	High	.45	.15	.15	.15	.15	.15	.15
	Low	.14	.14	.14	.14	.14	.14	.14
Barium chloride (short ton)	High	115.00	100.00	95.00	72.50	66.00	65.00	$70.00
	Low	50.00	80.00	70.00	60.00	60.00	52.00	54.00
Barium nitrate (pound)	High	.10¼	.10¼	.10	.09	.09	.10	.08½
	Low	.06	.07½	.08	.08	.07½	.07½	.08¼
Lithopone (pound)	High	.06¼	.07¼	.08 5/8	.06 5/8	.06 5/8	.05 7/8	.05 7/8
	Low	.06	.06	.06 3/8	.06 1/8	.05 7/8		.05¾

1/ Quotations on crude and ground barite are taken from Engineering and Mining Journal Press. Other quotations from Oil, Paint and Drug Reporter. The following are the designations, on which quotations in the preceding table are based:-

Crude barite, "f.o.b. mines."
Ground barite, "domestic, including barrels, f.o.b., St. Louis."
Lithopone, "in barrels, l.c.l."
Barium carbonate, "bags, works."
Barium chlorate, "kegs."
Barium chloride, "crystals, bags."
Barium nitrate, "casks."
Barium sulphate (blanc fixe), "dry, in barrels, l.c.l."
Barium sulphate (blanc fixe), "pulp works."

The following is a list of producers of ground barite and barium products:

GROUND BARITE

Name and Address of Company	Location of Plant
Chemical & Pigment Co. (Inc.), Oakland, Calif.	Oakland, Calif.
Clinchfield Sand & Feldspar Co., Baltimore, Md.	Kings Creek, S. C.
C. P. DeLore Co., St. Louis, Mo.	Miss. River & River Des Peres, St. Louis, Mo.
Flynt Silica & Spar Co., Los Angeles, Calif.	Daggett, Calif.
National Pigments & Chemical Co., St. Louis, Mo.	St. Louis, Mo.
Thompson, Weinman & Co., New York, N. Y.	Cartersville, Ga.

LITHOPONE

Name and Address of Company	Location of Plant
Chemical & Pigment Co. (Inc.), Oakland, Calif.	Oakland, Calif.
Chemical & Pigment Co. (Inc.), Oakland, Calif.	Collinsville, Ill.
Chemical & Pigment Co. (Inc.), Oakland, Calif.	St. Helena, Md.
E. I. du Pont de Nemours & Co., Wilmington, Del.	Newark, N. J.
E. I. du Pont de Nemours & Co., Wilmington, Del.	Philadelphia, Pa.
Eagle-Picher Lead Co., Chicago, Ill.	Argo, Ill.
Grasselli Chemical Co., Cleveland, Ohio.	Grasselli, N. J.
Krebs Pigment & Chemical Co., Newport, Del.	Newport, Del.
Mineral Paint Zinc Co., New York, N. Y.	Depue, Ill.
New Jersey Zinc Co. (of Pennsylvania), New York, N. Y.	Palmerton, Pa. (2 plants)
Sherwin-Williams, Cleveland, Ohio.	Chicago, Ill.
United Color & Pigment Co., Newark, N. J.	Newark, N. J.

BARIUM CHEMICALS[5/]

Name and Address of Company	Location of Plant
Alton Barium Products Co., Alton, Ill.	Alton, Ill.
Barium Products (Inc.), Modesto, Calif.	Modesto, Calif.
Barium Reduction Corporation, Charleston, W. Va.	So. Charleston, W. Va.
Chicago Copper & Chemical Co., Chicago, Ill.	Blue Island, Ill.
E. I. du Pont de Nemours & Co., Wilmington, Del.	Philadelphia, Pa.
Grasselli Chemical Co., Cleveland, Ohio.	Grasselli, N. J.
Oakland Chemical Co., New York, N. Y.	New York, N. Y.
Titanium Pigment Co. (Inc.), New York, N. Y.	Carondelet Station, St. Louis, Mo.

5/ Includes only those using barite.

U. of I. DUPS.
1942

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DECEMBER, 1929

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

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NO. VIII

BY

FREDERICK W. LEE

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GEOPHYSICAL ABSTRACTS[1]

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Compiled by Frederick W. Lee[2]

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1. GRAVITATIONAL METHOD

DIE REDUKTION DER KOINZIDENZZEITEN VON PENDELN ZUR BERECHNUNG VON SCHWEREDIFFERENZEN

(CORRECTION OF COINCIDENCE MOMENTS OF PENDULUMS FOR THE CALCULATION OF GRAVITY DIFFERENCES)

By H. Schmehl

Zeitschrift fuer Geophysik, Heft I, 1929, pp. 1-16.

The most common proceedings for calculating the correction of gravity observations made by pendulums such as the determination of the observed time of oscillations of a pendulum from the observed coincidence moments, the reduction of the observed times of oscillation to equal ratios, and the calculation of the relative gravity from the reduced differences of the time of oscillations, are replaced by reduction of the observed coincidence moments to equal ratios and calculation of relative gravity from the reduced differences of the coincidence moments. The reduction formulas are usually derived for the reduction of observations of 1/i second pendulums. In addition to the one-pendulum proceeding, the two-pendulum proceeding is taken into consideration also. Numerical examples serve for the explanation of the correction.--Translation of the author's abstract.

BEITRAG ZU FRAGE DER BEWERTUNG DER VERSCHIEDENEN SCHWERESTOERUNGEN

(CONTRIBUTION TO THE QUESTION OF THE ESTIMATION OF VARIOUS GRAVITY DISTURBANCES)

By Johannes Wilhelm

Abhandlungen der Preussischen Geologischen Landesanstalt, Berlin, Heft 10, 1928, 50 pp.

The fact that the evident mass anomalies in the subsoil are more or less compensated by isostasy is at the present time indisputable. This distribution of masses is in general proved by gravity measurements. There are two hypotheses by which the manner of compensation is explained: Pratt's hypothesis and Airy's hypothesis. Attempts based on gravity measurements to reach a common definition of the hydrostatic system, as given by Pratt or Airy, have failed. To make a decision between Pratt and Airy based on gravity measurements only seems to be impossible at the present time.

The correctness of the isostatic method of reduction as concerns the exact definition of the hydrostatic system, however, must be considered to be no higher than that of ordinary methods of reduction.

Defects of the isostatic method of reduction depend in the first place on the number and kind of hypothetical assumptions. The importance of the Bouguer anomaly is still maintained, although the direction is different. The diagnosis of the conditions of the masses in the uppermost zones at the earth's sectors is completed by geological interpretation of the gravity

anomalies. From this the value of the gravity disturbances necessary for the geologist must be derived. It results that the gravity anomalies provide for the geologist indications on the regional conditions of the masses at great depths on the largest tectonic unities. Local mass disturbances, observed very close to the surface, which can directly be investigated in a geological way, are not involved in the gravity measurements. This circumstance must be considered to be of great advantage. The gravity measurement is very suitable for the first preparatory work necessary for the systematic economic exploration of deposits of large areas.

A long list of literature is added.--Author's abstract, translated by W. Ayvazoglou.

COMPARISON OF TORSION BALANCE AND GEOLOGIC INTERPRETATIONS OF OUTLINE OF BLUE RIDGE SALT DOME

By Jack Logan

The Oil Weekly, Vol. 55, No. 4, Oct. 11, 1929, pp. 37-41.

After mentioning a series of oil discoveries made with the torsion balance in the Gulf Coast fields since 1923, and dwelling especially on the three commercial wells completed since August 15, 1929--the Esperon Domes and the Mykawa well, the first two daily producing 400 barrels of oil from a depth of 3,304 to 3,320 feet and 175 barrels from a depth of 2,270 to 2,276 feet, and the third producing gas from a depth of 4,189 feet--Logan describes the torsion balance survey of the Blue Ridge Dome, Fort Bend County. The results of the work are shown in curves and maps. A graphic comparison of the torsion balance and geologic interpretations of the outline of the dome is given.

The torsion balance survey was made in 1925 by the North American Exploration Co. The results of that torsion balance survey were worked into the accompanying drawings along with the geologic section by John F. Weinzierl of Houston. A map of leases and ownerships with the producing wells around the dome, forming a near-circle, is given.

The dotted curve shows the outline of the dome as revealed through the torsion balance survey. The torsion balance outline shown on the ownership map is a projection on it of the gradient map shown in another figure.

Figures showing "gradient curves" and "geologic sections" reveal the close conformity between the shape of the dome as shown by the logs of wells and through the torsion balance work.

The torsion balance interpretation of the outline of the dome conforms with startling accuracy to the geologic interpretation.

A brief description of the torsion balance work and the principles upon which it is based are given in the article.--W. Ayvazoglou.

ASTRONOMERS COMPLETE GRAVITY COMPARISON

Editorial Note

Canadian Mining Journal, No. 41, Oct. 11, 1929, p. 960.

A series of pendulum observations at base stations in Europe and America has been completed by the Dominion Observatory in Ottawa. This work is considered to be the most accurate gravity comparison ever made between the two continents.

The selected stations were: The Dominion Observatory, Ottawa, which is the gravity base for Canada; the Royal Observatory, Greenwich, the base for Great Britain; the Geodetic Institute at Potsdam, Germany, the base station to which all gravity stations throughout the world are referred; and the Coast and Geogetic Survey Office, Washington, the base for the United States.

The purpose of the work was to determine the gravity relation that exists between Ottawa and the bases mentioned.

From the scientific point of view, the results of the work are important in connection with the question of the shape of the earth as well as of the general equilibrium of the earth's crust.

According to the results of the determinations, the general equilibrium is found to be more complete and the equator more nearly circular than previous international gravity observations would indicate.

From the practical point of view the determination of gravity has been found to be important in the discovery of minerals and the tracing of geological structure.

The determinations were made by allowing a pendulum to swing freely in an air-exhausted chamber for a period of two weeks and determining accurately at each place how much the pendulum lost or gained. It was found that on taking the pendulum from Ottawa to Greenwich the pendulum gained about 2 minutes and 50 seconds per week; at Potsdam, about 3 minutes and 42 seconds; at Washington it lost about 2 minutes and 36 seconds in the same time. The comparative values of gravity at the four bases have been determined from these results.--W. Ayvazoglou.

2. MAGNETIC METHODS

CONTROL AND ADJUSTMENT OF SURVEYS WITH THE MAGNETOMETER OR THE TORSION BALANCE

By Donald C. Barton

Bulletin of the American Association of petroleum geologists, vol. 13, No. 9, September, 1929, pp. 1163-1187.

Some degree of residual error is present in practically every observation made with a magnetometer or a torsion balance. Errors of some types

can be avoided, or greatly reduced, by careful technique of observation. Other errors can be reduced by the application of corrections, but in general some residual error will persist. Errors in isolated or widely separated observations, or, in general, in single lines of stations, are not detectable or at least are not easily evident. But if the stations are so placed that there are closed circuits of observations, the errors generally produce more or less discordance.

Adjustment of mutually discordant systems, or sets of stations, is necessary for their satisfactory use for all but the rougher interpretations. The adjustment gives a probable value for each station with due regard to all of the observations more or less mutually affecting it. Various types of control and adjustment are in use. The method of least-square adjustment is the most accurate of the methods of adjustment, but requires at least moderate mathematical skill and is distinctly tedious. But for the degree of accuracy in commercial geophysical work in oil geology, certain approximate schemes of least-square adjustment can be cast in simple tabular forms which are easy to use and which require no knowledge of higher mathematics.--Author's abstract.

SIMPLE MAGNETIC METHOD FOR ORE PROSPECTING

By Hans Lundberg

Canadian Mining and Metallurgical Bulletin, July, 1929, 9 pp.

The present article is an attempt to help the prospector regain interest in the magnetic instruments and to show the use and application of a simple magnetic survey; most articles written on magnetic methods of prospecting are generally too elaborate and technical for the average prospector and thus have caused a great deal of disappointment.

Simple methods are investigated and an analysis is made of the results obtained by these methods.--W. Ayvazoglou.

THE VARIATION OF MAGNETIC ANOMALIES

By W. N. McFarland

Terrestrial Magnetism and Atmospheric Electricity,
vol. 34, No. 2, June, 1929, pp. 143-146.

This paper describes briefly a method of isogonic-chart construction, and the distribution of station anomalies, or differences of observed values from the average condition shown by the chart. The distribution is shown to be according to the law of occurrence of accidental errors. The available data are given in a table, and a figure shows the correspondence of observational data and theoretical curve in an individual case.--Author's abstract.

THE MAGNETO-CHRONOGRAPH AND ITS APPLICATION TO MAGNETIC MEASUREMENTS

By H. E. McComb and C. Huff

Terrestrial Magnetism and Atmospheric Electricity,
vol. 34, No. 2, June, 1929, pp. 123-141.

The period of a magnet oscillating under the directive force of the earth's magnetic field is a function of the strength of that field. Heretofore the estimation of the period has been accomplished by eye and ear readings, but in the present method the personal equation is entirely eliminated. Light from an intense source such as an automobile headlight lamp is condensed on the glass-scale end of a magnetometer magnet, and afterward reflection is focused on a slit in front of a photo electric cell. The reflected beam acts as an optical lever whose period is the same as that of the magnet but whose amplitude is twice as great. At each transit of the image across the slit, current is passed by the cell, and after passing through amplifiers is made to operate a pen on a chronograph.

A chronometer in the circuit marks seconds on the record by the same pen. In this investigation the method was applied in the determination of the moment of inertia of a magnotometer magnet, estimation of personal equation in eye-and-ear readings, and in preliminary tests of the relative absolute magnetometer.--Author's abstract.

THE MAGNETIC FIELD OF BODIES OF REGULAR SHAPE FROM THE VIEWPOINT OF MAGNETOMETRIC STUDIES

(In Russian)

By J. Bahurin

Bulletin of the Institute of Practical Geophysics,
Leningrad, No. 4, 1928, pp. 1-79.

This paper is a continuation of the article printed in issues Nos. 2 and 3 of the same bulletin (See Geophys. Abs. 1).

Chapter III deals with the "Oblate ellipsoid of revolution magnetized by forces not parallel to the axis of rotation (oblique magnetization)." The author concludes from the theoretical calculations that an oblate ellipsoid of rotation and an elliptical cylinder with an infinitely long axis are magnetized equally if the sections along the axis of rotation of the ellipsoid and that perpendicular to the axis of the cylinder are similar.

In the further part of this chapter Bahurin examines the questions and derives formulas concerning the following items:

1. The distribution of the magnetism along the surface of the ellipsoid.

2. The value of the horizontal component of the intensity of magnetization.

3. The potential and the component forces of the action of the oblate ellipsoid of rotation in case of oblique magnetization.

4. The changes of the horizontal component in the XOZ plane (in case of negative X and positive X).

5. The changes of the vertical component in the XOZ plane (in case of positive X and negative X).

6. The changes of the vertical and horizontal components in the ZOY plane.

7. The triaxial ellipsoid.

Many tables are included in the article.--W. Ayvazoglou.

ANOMALIES OF VERTICAL INTENSITY COMPARED WITH REGIONAL GEOLOGY FOR THE STATE OF CALIFORNIA

By George B. Somers

The Colorado School of Mines Magazine,
Sept., 1929, pp. 22-28.

The article is a preliminary report prepared by the author for the purpose of obtaining information regarding methods of procedure and the possibilities of utilizing the data available for a thesis on the correlation of the anomalies of vertical intensity with the regional geology of North America.

After a brief description of the morphological conditions in California and of the statigraphy and formations of rocks, the author gives an outline of the geological history of the following regions: 1. Basin and range; 2. Sierra Nevada; 3. California valley, and 4. Coast range region.

The source of information of the magnetic stations in California was the map of the western half of the United States which showed the vertical intensity of the magnetic stations of the region based on data obtained in 1925.

As this map gave only the actual vertical intensity at each station, in order to show the true anomalies it was necessary to make a correction for the changes in vertical intensity due to changes in latitude and longitude.

The method of doing this, selected by the author, is described. A table showing the magnetic correction for latitude and longitude for 158 stations is given. The conclusions drawn by Somers are summed up as follows:

1. The method used in eliminating the natural increase in vertical intensity from south to north and from west to east was satisfactory for this State, though it probably would not be for a larger area. The anomalies were of such magnitude that errors of considerable size would not have changed the results.

2. The regional structure appears to have been brought out to a considerable extent but would undoubtedly have been shown better if the results could have been obtained from a larger number of stations.

3. The producing oil fields are not indicated by the isonomalies, due probably to a shortage in the number of stations.

4. Since similar magnetic and structural conditions appear to exist to the north of the present fields, the possibility of additional fields in that direction is promising.

5. Assuming that the stratigraphy is similar to that in the producing areas, the indications appear to be favorable for additional oil pools along the center line of the California valley.

6. It is once more shown that the correct interpretation of geophysical data depends to a great extent on the study of geological conditions.

7. It is possible to obtain considerable data regarding the regional geology by geophysical methods.

A map showing the isonomalics of vertical intensity of the State of California is given.--W. Ayvazoglou.

MAGNETIC SURVEY IN THE REGION OF NEFTE-CHALA

(In Russian)

By A. J. Zaborovsky

The Petroleum Industry (Moscow),
vol. 16, No. 5, May, 1929, pp. 643-654.

Magnetic anomalies in the regions of oil deposits in Azerbeidjan were established by investigations made during 1926, 1927, and 1928.

These anomalies were found to be due to the presence of magnetic minerals, especially magnetite, in the rocks composing the oil-bearing regions. Zaborovsky gives a table of the distribution of the magnetic minerals in the region examined. The figures of the table show the relative content of the magnetic minerals in the rocks; their absolute amount in the rocks is very small, about 1 per cent on the average, although this amount increases in some places to 35 per cent.

In order to establish the geological structure of the region in places where the direct geologic investigation could not be made, the presence of this magnetic mineral was utilized for accomplishing the magnetic survey. The author describes the two types of instruments used in modern magnetometry for studying anomalies of the order from 100 to 200 γ: (1) The magnetic theodolite and a sensitive dip circle provided with a needle, or an induction

dip circle, and (2) The local variometer of various construction.

After comparing the advantages and disadvantages of the work with one or the other type of instruments, the author expresses himself in favor of the work with the local variometer and gives a description of the principles, construction, and the use of different types of variometers.

An expedition was formed by the Governmental Oil Research Institute for a detailed magnetic survey in the region of Nefte-Chala in 1928. The expedition was provided with four variometers, two for the determination of the horizontal intensity and two for the determination of the vertical intensity. The instruments were adjusted in accordance with the conditions of the work in the region.

The region of Nefte-Chala is situated near the mouth of the river Kura on the peninsula bounded on one side by the Caspian Sea and on the other by the Kisil-Agach Bay (Caucasus).

The preliminary magnetic investigation performed in 1927 has shown that the zone of magnetic anomalies runs in the direction from southeast to northwest.

Zaborovsky gives a series of diagrams and maps showing the distribution of the components of the magnetic field, on which he bases his summary of the results of the magnetometric survey of the region, as follows:

1. The magnetic anomalies in the region of Nefte-Chala are in strict relation with the geological structure of the region;

2. The character of these anomalies, if they are caused by the presence of magnetite in the Tertiary deposits, shows that:

    A. There is in this region an anticline extending in the direction from southeast to northwest;

    B. The line of the axis of the anticline passes to the east of pit-hole 5;

    C. The arch of the anticline sinks gradually toward the southeast;

    D. The northeastern wing of the anticline has a smaller angle of dip than its southwestern wing; and

    E. To the southeast of the line of pit-holes the arch of the anticline rises again. -- W. Ayvazoglou.

3. SEISMIC METHODS

GEOPHYSICAL FOUNDATION STUDY BY EXPLOSION-WAVE METHOD

By A. T. Parsons

Engineering News-Record, Feb. 14, 1929, pp. 273-275.

Parsons mentions the possibilities of applying seismic methods of geophysical prospecting in making foundation explorations at proposed sites for dams, bridges, or buildings.

The explosion-wave method, as it is developed in oil-field work, is described by the author in the article. Several figures showing typical time-travel record and explosion-wave paths from single and successive shots are given.

During the past two years Parson's organization has made subsurface foundation surveys of several hundred square miles in various parts of California. Where it has been possible to compare the results with actual drilling, a close agreement has been observed.--W. Ayvazoglou.

UBER FORTPFLANZUNG VON ELASTISCHEN WELLEN IN VISKOSEN MEDIEN

(ON THE PROPAGATION OF ELASTIC WAVES IN VISCOUS MEDIUMS)

By B. Gutenberg

Physikalische Zeitschrift, vol. 30, No. 8, 1929, pp. 230-231.

According to the theoretical conclusions made by Sezawa, elastic waves become gradually longer and more level during their propagation in viscose mediums. The author finds that under certain suppositions the law of propogation may be written as follows:

Period $T = \sqrt{T_0^2 + CD}$ in which D is the distance and C the constant for the material in question.

This relation has proved to be correct for a series of geophysical proceedings, especially for the propagation of waves of seismic disturbances of the soil, earthquakes, artificial shakings, sea waves, and also correct qualitatively for sound waves, where quantitative results are not available.--Author's abstract.

4. ELECTRICAL METHODS

ELECTRICAL SUBSOIL EXPLORATION AND THE CIVIL ENGINEER

By Irving B. Crosby and Sherwin F. Kelly

Engineering News-Record, Feb. 14, 1929, pp. 270-273.

The adaptation of modern geophysical procedures to important requirements of civil engineering construction is outlined.

The authors describe the practical application of electrical exploration to certain foundation problems, demonstrating how effectively it may be used for the investigation of bedrock concealed by overburden.

The demonstration of the method was arranged at eight preexisting borings, and the electrical observer was kept in ignorance of the conditions already encountered in the course of drilling. The results were checked with the drill records and found to be satisfactory.

The results of some of the determinations are described in the article as representative examples. Various graphs show the resistivities calculated as a function of depth.

The average accuracy in 10 of the electrical determinations that were directly checked by the drill is shown in the table given below:

Table 1. - Results of electrical determinations directly checked by the drill

Hole No.	True depth to rock, feet	Predicted depth, feet	Predicted depth in relation to the true depth, per cent
1	29	24	83
7	27	41	151
9	108	108	100
10	100	118	118
11	56	53	95
12	50	43	86
13	101	105	104
14	142	145	104
20	167	115	69
21	147	142	97

Following the tests an extensive exploration campaign was undertaken. The results are not made public, but the campaign provides a basis for cost estimates.

Two areas, totaling 250 acres, were surveyed by establishing about 200 points of observation.

From one to three determinations could be made at a cost of $50 to $100 per position.

The total cost of the survey to the construction company was not much more than that of one drill hole in which much trouble was caused by bowlders.

The authors conclude that the example cited proves the possibility of the application of electrical methods to a specific foundation problem.

As far as the authors are aware, this was the first time that the geophysical method of prospecting has been commercially applied in solving civil engineering problems.

The geophysical method of prospecting was applied last spring to the study of two dam sites on the Connecticut River near Littleton, N. H.--A. Ayvazoglou.

EQUIPOTENTIAL LINES OF A NATURAL ELECTRIC FIELD PRODUCED BY A SPHERICAL ORE-BODY

(In Russian)

By A. Petrowsky

Bulletin of the Institute of Practical Geophysics, Leningrad, No. 4, 1928, pp. 81-89.

In two previous articles, "Natural electric field produced by ore" and "Determination of the location, depth, and thickness of a spherical ore body by observing the earth current produced" (see Geophys. Abs. 1), Petrowsky constructed graphs showing the changes of the potential gradient which occurred in case the observer was moving across the region situated above the ore body.

In addition to these graphs the present article deals with methods of calculation and construction of graphs showing the equipotential lines produced on the earth's surface due to the action of a spherical ore body.

Calculation is made for three cases:

1. For the vertical position of the axis of polarization ($\alpha = 0^\circ$).

2. For an inclined position of the axis of polarization ($\alpha = 45^\circ$).

3. For the horizontal position of the axis of polarization ($\alpha = 90^\circ$).

According to these graphs:

In the first case ($\alpha = 0^\circ$), the equipotential lines of a natural electric field of a spherical ore body are circles, the center of which lies above the center of the ore body.

In the second case ($\alpha = 45^\circ$), a deformation of the curves takes place; the maximum of the potential shifts somewhat forward and the positive equipotential lines are shifted forward at the same time.

In the third case ($\alpha = 90^\circ$), the position of the equipotential lines becomes symmetrical with respect to the axis of ordinates.

The corresponding tables used for the construction of the graphs are added to the article.--W. Ayvazoglou.

MAGNETIC FORCES IN AN ARTIFICIAL ELECTRIC FIELD

(In Russian)

By A. Petrowsky

Bulletin of the Institute of Practical Geophysics, Leningrad, No. 4, 1928, pp. 92 - 119.

In his article "Calculations of an artificial electric field" (see Geophys. Abs. 1), Petrowsky derived formulas for the calculation of potentials and projections of the electric force and showed the methods for calculating them. The present article presents formulas for calculating projections of the magnetic force, as well as the corresponding methods of these calculations.

The mathematical analysis of a magnetic field produced on the surface of the earth by leading-in of the current through one electrode has already been given by Koenigsberger (Das magnetische Feld einer Stromquelle im Raum, Physikal. Ztschr., vol. 28, No. 9, 1927, pp. 342-344. Petrowsky examines a more general case, that of a magnetic field produced at any point in space, the system of coordinates being changed accordingly.

Using the principle of superposition of fields, Petrowsky derived formulas by which the magnetic force on the surface of the earth in Schlumberger's, Petrowsky's and Lundberg's fields could be calculated.

The last part of the article gives the methods of calculation of the magnetic forces in Petrowsky's field.

Four numerical examples, completed by two tables, illustrate the proceeding of the calculation. The use of the tables considerably shortens the time necessary for calculations. The article is provided with five figures.--W. Ayvazoglou.

ELECTROMETRIC INVESTIGATION OF THE UPPER ARSHINSK ORE-BED ACCOMPLISHED IN THE SUMMER OF 1927

(In Russian)

By A. Petrovsky

Bulletin of the Institute of Practical Geophysics, Leningrad, No. 4, 1928, pp. 121 - 143.

Valuable results obtained from the electrometric investigation of the Upper-Arshinsk ore bed accomplished in the summer of 1926 (see Geophys. Abs. 1) caused the continuation of the work in the summer of 1927.

Contents of the article:

1. The object of the expedition.

2. Preparation for the expedition.

3. Changes made in the stations.

4. Changes concerning the feeding network.

5. Auxiliary instruments.

6. Changes in the receiving devices.

7. A field bridge for resistance measurement.

8. Electrodes of the bridge and the method of measuring resistances of the rocks on the spot.

9. Area of the work.

10. Cutting of the area into sections.

11. Electrical surveying.

12. Classification of the anomalies. Importance of the specific conductivity of the rocks.

13. Influence of topography.

14. Influence of meteorological conditions.

15. Conclusion.

Work on the following details was included in the plan:

A. Definition of the changes in the course of the equipotential lines caused by the difference of the specific conductivity of the upper layers of the soil.

B. Establishment of topographic influence upon the course of the equipotential lines.

Changes made with regard to generator sets, rods, arrangement of the field bridge, as well as the new bridge itself, are given in figures.

The work of the expedition began from the last station of the previous year, and new stations were added by extension.

The position of the stations is given in a figure. A considerable amount of numerical data has been obtained, but the working out of it is of course delayed, as supplementary investigations in the laboratory are required. However, the authors give in the article several conclusions based on the preliminary examination of the data; they are:

1. The electric field observed in field work resembles, on a whole, a field calculated theoretically.

2. Although the fields are always somewhat distorted, the varieties of these distortions may, to a certain degree, be systematized into groups of wavy, slanting, and curved fields.

After a description of the influence of topography and meteorological conditions, the authors conclude that a careful taking into consideration of all kinds of data obtained during the electrical research is necessary, and that it is indispensable, in addition to the survey of the equipotential lines even accompanied by the measurement of the gradient, to accomplish a supplementary work by measuring the specific resistivity of the upper layers of the soil, the specific resistivity of rocks and other layers which break the homogeneity of the field, as well as of the water in the swamps and basins scattered in the area under investigation. Levelling has, according to the authors, only secondary significance, but the influence of meteorological conditions under which the research work is performed is most important.--W. Ayvazoglou.

AN ARTIFICIAL ELECTRIC FIELD WITH 21 PAIRS OF ELECTRODES

(In Russian)

By A. Petrowsky

Bulletin of the Institute of Practical Geophysics, Leningrad, No. 4, 1928, pp. 216-231.

Contents of the article:

1. General information.
2. Calculation of the potential.
3. Calculation of the equipotential lines.
4. Calculation of the longitudinal component of the electric force.
5. Calculation of the transversal component of the electric force.
6. Calculation of the value and direction of the total electric force.
7. Calculation of the longitudinal component of the magnetic force.
8. Calculation of the transversal component of the magnetic force.
9. Calculation of the value of the total magnetic force.
10. Calculation of the direction of the magnetic force.

In one of his previous articles, "Calculation of an artificial electric field," Petrowsky derived formulas and gave detailed schemes of the calculation of an artificial electric field. The present article examines the calculation of a square field having 21 electrodes on each of the two opposite sides.

The results of these calculations are disposed of in eight tables as follows:

I. A table of potential numbers.

II. A table of relative coordinates of the equipotential lines.

III. A table of longitudinal numbers of force.

IV. A table of transversal numbers of force.

V. A table of longitudinal magnetic numbers.

VI. A table of transversal magnetic numbers.

VII. A table of total magnetic numbers.

VIII. A table containing the angles of inclination of magnetic force to the direction of the lines of the field.

The eight figures given in the article represent:

1. The distribution of the coordinate axes and the numbering of lines and electrodes.

2. Theoretical distribution of equipotential lines in a square field with 21 pairs of electrodes.

3. A graph of longitudinal numbers of force.

4. A graph of transversal numbers of force.

5. A graph of longitudinal magnetic numbers of a square field with 21 pairs of electrodes.

6. A graph of transversal magnetic numbers.

7. A graph of the total magnetic numbers.

8. Theoretical distribution of the magnetic force in a square field with 21 pairs of electrodes.

In the continuation of this article, the author expects to give tables which will make possible the calculation of electrical and magnetic fields not only in the planes of the electrodes but also in space.

These tables will probably appear in the next issue (No. 5, 1929) of the Bulletin of the Institute of Practical Geophysics.--W. Ayvazoglou.

UBER EINIGE GEOELEKRISCHE AUFNAHMEN AM RAMMELSBERG UND IM OBERHARZ

(CONCERNING SOME GEOELECTRIC MEASUREMENTS ON THE RAMMELSBERG AND IN UPPER HARZ)

By Dr. A. Ebert

Metall und Erz, vol. 18, Sept., 1929, pp. 462-467.

The author compares geoelectric methods of prospecting with some others, especially with the gravimetric, magnetometric, and seismic methods, noticing briefly the advantages and disadvantages of each.

In the second part of the article Ebert describes the experiments with electric methods of prospecting, performed under the auspices of the recently created section of the Geological Division of the Prussian Geological Institution (Preussische Geologische Landesanstalt.)

The well-known ore deposit regions of the Rammelsberg and Upper Harz were chosen for the experiments.

The inductive method has been tried out especially, and the results of investigations are given in a series of maps and diagrams.--W. Ayvazoglou.

5. RADIOACTIVE METHODS

ON THE METHODS OF MEASUREMENT OF SUBSTANCES POSSESSING WEAK RADIOACTIVITY BY MEANS OF α-RAYS

(In Russian)

By V. Baranov

Vestnik du comité géologique, leningrad, vol. 2, No. 7, 1928, pp. 37-41.

The measurement of radio elements in weakly radioactive ores (of the order of 0.01% U_3O_8 and less) by means of α-rays is sometimes very desirable.

In these measurements a quick method of work, as well as constancy of natural dissipation inside of apparatus, is important. In order to eliminate the defects of electroscopes of usual types used for these measurements, Baranov constructed an apparatus by which a series of disadvantages could, to a certain degree, be avoided.

A schematical design of the apparatus is given in the article. The apparatus consists of a Wulf's electrometer provided with a cylindrical electrode 10 centimeters in length and 5 millimeters in diameter. The electrometer

is turned upside down and is secured to an ionization chamber; the latter has the shape of a cylinder 16 centimeters in diameter and 10 centimeters in height. The bottom of the chamber can be opened or closed hermetically.

In the center of the bottom of the chamber is soldered a pipe through which the air is pumped out of the chamber. The constancy of the air current is maintained by means of a special regulator and is controlled by a rheometer. The air enters the chamber through to symmetrically placed pipes 20 centimeters long and 1.3 centimeter in diameter; these pipes represent cylindrical condensers and are provided with internal electrodes 16 centimeters long and 5 millimeters in diameter insulated by means of ebonite plugs. The electrometer is connected with a battery of 200 Clark's elements. According to the scheme of this connection shown in a figure, the ionization chamber remains unloaded. This arrangement makes it possible to change the materials to be investigated without the necessity of disconnecting the battery.

Less than two minutes is required for the taking of each reading.

The advantages of the apparatus are as follows:

1. Owing to the fact that the apparatus remains charged and the potential difference at the insulator of the electrometer does not exceed 1 volt during the measurement, the polarization of the insulator is not an essential influence.

2. Owing to the uninterrupted flowing of the air current through the apparatus during the measuremont, the influence of the emanation of the material under investigation is greatly reduced.

3. For the same reason as given under (2) the influence of the ions in the air of the chamber is reduced and the change of the natural dissipation during the opening of the chamber is removed to a great degree.

4. The time necessary for taking measurements is reduced considerably, thus the importance of the fluctuations of natural dissipation from outside causes is lessened.

Several tables showing the results of observations made with this apparatus are given.--W. Ayvazoglou.

APPARATUS WITH CLOSED AIR CURRENT FOR THE DETERMINATION OF RADIOACTIVITY OF MINERALOGIC COLLECTIONS

(In Russian)

By S. A. Artsybyshew and V. Pleshanova

Vestnik du comité géologique, Leningrad,
vol. 3, No. 8, 1928, pp. 35-39.

After a brief notice on the different defects of apparatus used for determining the radioactivity of ores in mineral collections, the authors des-

cribe a new very simple apparatus constructed by them: A box containing the samples to be investigated is put over a zinc sheet and covered with a square metallic chamber (cover) 55 by 55 by 27 centimeters in size. An electroscope, provided with a reading microscope, is fixed to the chamber; the electroscope is connected to a commutator in the form of a square wire frame. The upper part of the chamber has an opening through which a propeller is inserted; the propeller is put in rotation by means of an electromotor. Owing to this arrangement, a strong air current is obtained inside of the chamber, so that all dead spaced are climinated.

This arrangement also makes it possible to reduce the size of the commutator and to raise it considerably above the bottom of the chamber; the capacity of the device can be reduced and becomes independent of the size of the samples to be investigated.

The capacity of the apparatus described is constant (with an accuracy of about 5%) and is equal to 20.7 centimeters in case a double wire-frame commutator is used, and to 13.6 centimeters for a single wire-frame commutator; the size of the commutator is 25 by 25 centimeters.

Four disks, each 5 centimeters in diameter, coated with a thin layer of U_3O_8 were used. The disks and the samples were distributed over the zinc sheet arbitrarily.

The distribution of the disks and samples is shown in diagrams, and the data obtained accordingly are given in tables.

The authors proved also that an active sample may be discovered easily by successive covering of the minerals with a piece of heavy cloth.

The results of the experiments are considered to be successful.--W. Ayvazoglou.

GRADUATION OF APPARATUS FOR THE MEASUREMENT OF RADIOACTIVITY

(In Russian)

By A. A. Lomakin

Bulletin of the Institute of Practical Geophysics, Leningrad, No. 4; 1928, pp. 151-156.

In the present article Lomakin describes a simple and sufficiently accurate method for comparison of two capacities, one of which is the capacity of an α- electroscope of the usual type.

The method of measurement is given in a scheme.

The graduation of the scale of the apparatus in volts is made, based on the following calculations:

From the formula for the strength of ionization current, $i = \frac{C\,(V_1 - V_2)}{9.10^{11}\,t}$ amperes, in which the capacity C is given in centimeters and the voltage drops V_1-V_2 in volts, it is evident that for the calculations of the strength of current not its absolute tension is important, but the change of the tension, $V_1 - V_2$, during the measurement.

The interval of tensions on the scales of very sensitive instruments is small, while a full tension necessary for producing saturated current is more or less large. Therefore, a simple comparison of the reading of the electrometer with the reading of the voltmeter for the full tension V_1 is not rational.

A comparatively small error in the determination of the full tension may become of great value for the comparatively small difference $V_1 - V_2$.

The author eliminates these defects by measuring the tensions $V_1 - V_2$ by a second very sensitive voltmeter. Therefore, the error in the readings of the first voltmeter which shows the tension of the initial point of the apparatus scale remains constant for the whole graduation and does not influence the calculation of the value of the current strength.

The arrangement is shown in a scheme.--W. Ayvazoglou.

DEFINITION OF RADIUM IN RUSSIAN ORTHITES OF DIFFERENT ORIGIN

(In Russian)

By L. N. Bogoyavlensky

Bulletin of the Institute of Practical Geophysics, Leningrad, No. 4, 1928, pp. 157-164.

Bogoyavlensky analyzed three patterns of orthites from pegmatite veins of Russian origin but from different ore beds; the determination of the radium has been made by using the emanation method.

Radium has been found in all patterns, but its quantity could not be brought in connection with the content of thorite. As radium in orthite is in a state of radioactive equilibrium with uranium, the orthite has been calculated according to the quantity of uranium.

The author concludes that if the dependency of the content of alkaline metals on radium and uranium is taken into consideration, it can be supposed that the latter is present in orthite as a definite chemical combination and not as a microcosmic ingredient.--W. Ayvazoglou.

ANOMALIES OF THE PENETRATING EARTH RADIATIONS IN THE UKHTA OIL-BEARING REGION

(In Russian)

By L. N. Bogoyavlensky and A. A. Lomakin

Bulletin of the Institute of Practical Geophysics, Leningrad, No. 4, 1928, pp. 165-178.

The Ukhta oil-bearing region is situated in the basin of the Pechora River (North Russia) along the Ukhta River. The oil deposits occupy a vast area of several tens of square kilometers. In this region the rocks of the Devonian era form a great anticlinal fold. The analysis of Ukhta region brine has proved that it contained elements of radium (the highest content being 7.6×10^{-9} grams per liter). Investigations on the radioactivity of the deposits in the region were made in 1927. In addition to radium, mesothorium with all products of its dissociation was discovered.

It was found that the brine also contained protoxide combinations of iron which, in combination with the oxygen on the surface of the earth, were liberated in the form of fine ocher of different shades.

Investigations on penetrating earth radiations were made by means of a new electrometer.

The body of the apparatus used had walls 15 millimeters thick and the chamber could be closed hermetically. The volume of the closed chamber was 2,150 cubic centimeters. The authors investigated 44 points. Lead screens of a thickness up to 7.45 centimeters were used. The results of measurements are given in a table which shows that the intensity of radiation varied from 4.37 to 7 ions; that is, the variation reached 72 per cent. A map of radiometric survey based on the data obtained was prepared (given in a figure), and curves of equal intensities showing the values of 5.5, 6.0, 6.5, and 7 ions are drawn.

In the last part of the article the authors give the results of their study of radiation. They found that a point far from the accumulation of the radioactive sediments on the surface, the intensity of the radiations changed very little, and that the calculated coefficient of absorption was equal to 0.5, while the absolute values of the intensities were different. This indicated the insignificance of the quantity of soft radiations due to the radioactive upper strata of the soil.

The difference of absolute values of intensity at each point must, therefore, be assigned to the radiations penetrating from radioactive bodies situated at certain depths. The coefficient of absorption could not be determined by the authors; it diminished with the increase of the thickness of the lead screen.--W. Ayvazoglou.

I.C.6224.

RADIOACTIVITY OF ASHES OF SOME ROCK OILS

(In Russian)

By L. N. Bogoyavlensky

Bulletin of the Institute of Practical Geophysics, Leningrad, No. 4, 1928, pp. 311-314.

Bogoyavlensky describes the results of experiments performed by him to determine the radioactivity of ashes of rock oils. Samples from the Ukhta (North-Russia) oil-bearing region and the Maikop (Caucasus) region were taken. The choice of samples from regions so remote from one another was made because of the great difference of the radioactive phenomena of the brine of these two regions. While the brine from the Ukhta region contains a considerable amount of radium (up to 7.5×10^{-9} grams per liter), that of the Maikop region does not contain radioactive elements at all.

For the determination of the radioactivity of ashes about 200 grams of rock oil were slowly evaporated and the residue burned, weighed and measured in an α-electroscope. The activity of ashes is expressed in units of U_3O_8.

The results of the experiments may be seen from the following table:

ORIGIN OF THE SAMPLES		CONTENTS OF ASHES IN PER CENT	RADIOACTIVITY IN UNITS OF $U_3\ O_8$
Ukhta oil-bearing region	No. 1	0.03	0.014
	No. 2	0.098	0.003
Maikop region	No. 3	0.027	0.020

The figures show that the ash activity of oil samples proved to be in an inverse ratio to the contents of ashes. The analysis of ashes performed in order to discover the radioactive emanations showed a complete absence of radium and thorium X.

The author supposes that the activity of ashes is caused by the dissociation products of emanation (polonium) which are produced as a result of the great absorptive capacity of oil with relation to the radioactive emanations.

The experiments which are so far of a preliminary character, show, of course, that the active products are not connected with ashes but with the rock oil itself, as in the first case radium and thorium X would be discovered by the emanation method.--W. Ayvazoglou.

6. GEOTHERMAL METHODS

MÉMOIRE RELATIF À LA PRISE DE LA TEMPERATURE DANS LES SONDAGES

(NOTE CONCERNING THE TAKING OF MEASUREMENTS OF TEMPERATURE IN THE BOREHOLES)

By I. Atanasiu

Annales des mines de Roumanie,
No. 7-12, July-December, 1928, pp. 219-225.

The purpose of the present article is to find out the conditions under which the temperature measurements can be used as a means of determining occlusions produced by cementation.

The author divides the article into the following sections:

1. General considerations on the variation of the temperature in the ground.
2. What can be obtained by taking the measurements of temperature in the boreholes?
3. Conditions which the thermometers must satisfy.
4. Propositions for applying the measurements of temperature in boreholes.

In section one, Atanasiu gives a formula according to which a geothermic degree--that is, the number of meters at which the thermometer must be lowered in order to reach an increase of temperature of 1 degree--can be calculated: $Gr = \frac{h}{t-to}$. Here Gr is the geothermic degree, h the depth at which the temperature is measured, t the temperature obtained, and to the mean annual temperature of the ground.

In section two the author gives a table of measurements taken at different depths:

Oil Fields	Depth of Borehole	Temperature	Geothermic degree	Remarks
1. Filipesti	886	37.4°	32.6 m.	Without oil.
2. Campina	831	37.4°	30.5 m.	In boring.
3. Moinesti	900	33.7°	36.4 m.	Gas, water.
4. Luncacesti-Zames	480	21.5°	38.4 m.	Surface water.

If the occlusion is not good the water coming from above, which is colder, will cause the lowering of the temperature, and the geothermic degree calculated will accordingly show a higher value than the normal one for the region.

This is plainly shown in the table in the case of Luncacesti-Zames.

Temperature measurements may be useful for the determination of the origin of water appearing in the boreholes. For example, if the occlusion is no longer perfect, it is often difficult to be sure whether the water is originally in the deposit itself or comes from above.

By introducing a thermometer the source of the water can be established with surety, for the temperature will remain equal or be higher in the first case and will drop in the second case.

Section three deals with the construction and protection of thermometers for use measurements at great depths. The author proposes to manufacture the following types:

Type I. With a scale of from 10 to 30° for depths up to 500 meters.

Type II. With a scale of from 25 to 45° for depths from 500 to 1,000 meters.

Type III. With a scale of from 40 to 60° for depths from 1,000 to 1,500 meters.

In the last section Atanasiu proposes that temperature measurements be made obligatory; if taken as often as possible they will furnish valuable material which may serve for general orientation.--W. Apvazoglou.

DETERMINATION OF GEOTHERMAL GRADIENTS ON OIL STRUCTURES

By the American Petroleum Institute

Oil and Gas Journal,
Nov. 14, 1929, pp. 105-106.

This paper is a report of the research division of the American Petroleum Institute on the progress toward determining geothermal gradients in oil fields.

The report concerns the Oklahoma, Texas, Santa Fe, and Long Beach fields.

Oklahoma.-One of the most outstanding facts in connection with earth temperatures in Oklahoma is the apparent agreement of the dip of the isothermal surfaces and the formations.

The reciprocal gradients vary widely from area to area, but show small variations within the same area. The lowest gradient found was 1° F.

in 149 feet (Healdton); the highest, 1° F. in 36½ feet (Glenn Pool).

Data collected in two fields, Haverhill, Kans., and Glenn Pool, Okla., indicate that the temperatures are higher at corresponding depths in wells located in the producing area than in wells located outside the producing area.

Texas.-Temperature measurements were obtained from wells in all parts of the State. A complete report on the results of the work has been submitted to the American Petroleum Institute. The relationship between the normal rate of increase of temperature and the structure is especially noteworthy in salt domes and in anticlinal structures carrying much underground water.

Temperature measurements obtained in salt domes were especially interesting. The following results were obtained in the Humble Salt Dome, Harris County:

1. At equal depths temperatures are higher on the top of the dome than on the flanks.

2. Rate of increase in the temperature with depth is relatively greater on top of the dome than on the flank.

3. The difference in temperature at the same depth and the greater temperature gradient on top of the dome are found not only near the salt but also above the salt (at depths) of 100 and 250 feet).

4. Drill holes on top of the salt domes have approximately the same temperatures at equal depths.

Santa Fe Springs Field.-Fourteen wells were measured, bringing the total number tested to 33. The following conclusions were drawn from the computed data:

1. Depth temperature relations are irregular as compared with other fields;

2. Shallow temperatures vary within rather wide limits. Temperatures at a depth of 100 feet range from 65.9 to 74.9° F.

3. The elevations at which a temperature of 100° F. occurs define an irregular isothermal surface.

4. Reciprocal gradients computed over the depth interval 100 to 2,000 feet, vary from 47.5 to 56.6 feet per degree.

5. Reciprocal gradients recomputed over the same depth interval after eliminating certain abnormal temperatures, show a less pronounced irregularity.

6. The lack of a satisfactory relation between geothermal conditions and structure in this field may be attributed to the following three factors:

   a. The variation of the gradients within the narrow limits of only a few feet per degree of temperature;

   b. The unusual conditions of development and production involving the very close spacing of wells and the flow of large quantities of gas and oil under high pressure; and

   c. Natural conditions of ground-water circulation causing the establishment of abnormal shallow temperatures.

Long Beach Field.-Nineteen wells were measured in this field, making a total of 43 completed. The total variation of depth to a 100° isothermal surface is about 200 feet and of reciprocal gradients about 8 feet per degree. The mean isothermal depth is 1,534 feet and the mean reciprocal gradient about 53 feet per degree Fahrenheit.--W. Ayvazoglou.

TEMPERATURE GRADIENT IN THE PECHELBRONN OIL-BEARING REGION, LOWER ALSACE:

ITS DETERMINATION AND RELATION TO OIL RESERVES.

By I. O. Haas and C. R. Hoffmann

Bulletin of the American Association of Petroleum Geologists, vol. 13, No. 10, October, 1929, pp. 1257-1275.

The authors summarize their experience in making temperature measurements in oil wells and consider whether it is possible to ascertain the existence of petroleum on the basis of a smaller or a larger temperature gradient.

The article is divided into the following paragraphs:

1. Description and discussion of thermometers used.
2. Possible sources of error in temperature measurements.
3. Carrying out of temperature measurements.
4. Calculation of temperature gradient.
5. Criticism of results obtained.
6. Application of results obtained.
7. Attempt at an explanation of temperature variations.

In the abstract of the article given by the authors they say:

"Numerous temperature measurements have been made during the last 10 years in the Pechelbronn oil-bearing region, to aid in the search for petroleum deposits.

However, the results of the study of approximately 500 measurements clearly show that the isogeotherms are influenced primarily by the tectonic structure of the Rhine Valley graben. They show a nearly regular rise from the edges to the central part of the graben; also not uncommonly they approach one another. Such increase of temperature is especially marked in the fault zones.

Seemingly the isotherms are not influenced by larger or smaller oil accumulations; therefore, the presence of oil can not be looked upon as the cause of the relatively elevated temperature.

It is perhaps possible to explain the rise of temperature toward the central part of the graben by a gradual change in the nature of deposits. Coarse-grained sandy beds of the graben edges are gradually replaced toward the center by increasingly finer sands, and finally by finely laminated marls. It is possible that these finest deposits act as a sort of protective cover against the loss of heat. This suggestion would also explain the increase of temperature and the accumulation of heat along the faults as it is possible to suppose a stronger increase of temperature in the direction of the dislocation in the strata. This suggestion is strengthened by the fact that the especially high horsts are characterized by temperature maxima on the isothermal map."

A map of the Northern Pechelbronn oil fields showing the temperature curves at a depth of 400 meters and their relation to productive oil zones, as well as three cross sections of the oil fields showing underground temperature curves, are given.

A list of publications on temperature measurements is attached.--W. Ayvazoglou.

LA TEMPÉRATURE DES EAUX PROFONDES DE LA RÉGION PARISIENNE

(THE TEMPERATURE OF DEEP WATERS IN THE REGION OF PARIS)

By P. Lemoine and R. Nassans

Bulletin du muséum national d'histoire naturelle, series 2, vol. 1, No. 4, May, 1929, pp. 264-280.

In this article the authors sum up a series of data obtained on thermometrical measurements of deep waters in wells bored in the region of Paris. They consider these data to be of interest, notwithstanding the lack of certainty that the temperatures taken are of sufficient correctness.

After a brief description of the process by which the geothermic degree (Dg) should be calculated and the influence of the possible errors on its value established, the authors give some data concerning the varieties of the geothermic degree in different places, as determined by the following scientists:

Gensanne, who made experiments in 1740 near Belfort, found, according to Becquerel:

At a depth of 101 m. - - - -	12.5°	
Do. 206 m. - - - -	13.1°	Dg (101-206) = 17.5 m.
Do. 308 m. - - - -	19°	Dg (206-308) = 17.6 m.
Do. 433 m. - - - -	22.7°	Dg (308-433) = 33.8 m.

De Saussure (1785), who took measurements in Switzerland at wells bored in searching for rock salt, established these measurements:

At a depth of 108 m. - - - -	14.4°
Do. 183 m. - - - -	15.6°
Do. 220 m. - - - -	17.4°

Corder calculated the geothermic degrees (Dg) given for the following places:

Cormeaux - - - - - - - - - - - Dg = 36 m.
Littry - - - - - - - - - - - - Dg = 19 m.
Decize - - - - - - - - - - - - Dg = 15 m.

By comparing a series of data taken in the wells in Paris and its vicinity the authors found the mean value of the geothermic degree for the basin of Paris to be 34 meters.

Great anomalies were observed in Pressagny-l'Orgueilleux. The temperatures of waters at a depth of 70 meters were found to be 17°, although their normal temperature should be about 13°.

The authors note especially that, by assuming one geothermic degree to be equal to 34 meters, the waters of a temperature of 17° should be at a depth of about 204 meters, but none of the wells situated in the region to the northwest of Pressagny was so deep as that.

Whether this anomaly can be explained by the quick rising of deep waters flowing from the region of Paris to Pressagny remains unsolved, and the conclusion of the authors that the waters in Pressagny are supplied from south and east seems to be the most probable one.

Anomalies observed in several other wells were less important.

In several places waters of a temperature lower than the normal one have been observed. The authors, who called these waters "hypothermal" waters, could not find any explanation for this hypothermality.

A long list of authors giving the data for the temperatures at different places is added to the article.--W. Ayvazoglou.

7. UNCLASSIFIED METHODS

LES METHODES GEOPHYSIQUES DE PROSPECTION

(GEOPHYSICAL METHODS OF PROSPECTING)

By Georges-Albert Boutry

La revue pétrolifiere, Paris,
No. 325, June 15, 1929, pp. 821-825.

In this introductory article Boutry examines the question of geophysical methods of prospecting, especially with regard to prospecting for oil.

A brief discussion of the following subjects is included:

1. Hypotheses on the origin of oil.
2. Increasing difficulties for prospecting oil.
3. Oil-bearing regions of the world.
4. Principal favorable structures of the terrane; and
5. Geological prospecting and its insufficiency.

After giving a general outline of the principles of the methods and their application the author concludes:

"I will try to show that, although the efficiency of geophysical methods of investigation has often been discredited or, on the contrary, exaggerated, it remains true that geophysical prospecting has high value and power and that the use of these methods will become in the very near future absolutely necessary for all kinds of prospecting by which a success is to be obtained."

This introductory article is followed by a series of articles, the purpose of which is to show that the problems for the research of salt domes, faults of monocline structure, anticlines, etc., can be solved by geophysical methods; the possible degree of the solution is examined.

The whole work is arranged according to the following classification:

I. Examination of gravity anomalies: pendular method, torsion balance method.

II. Examination of magnetic anomalies.

III. Examination of electrical anomalies by means of direct current and alternating current.

IV. Examination of elastic anomalies; study of the velocity of propagation of shocks; measurement by sound; measurement by ultrasound.

V. Anomalies of geothermal gradient. Propagation of radiations of short waves in the rocks. Radioactivity in oil areas.

All the articles are provided with many figures and diagrams and a long list of literature is given.--W. Ayvazoglou.

GEOPHYSIKALISCHE ZUSAMMENHÄNGE ZWISCHEN OSTALPEN UND BÖMISCHER MASSE

(GEOPHYSICAL RELATION BETWEEN THE EAST ALPS AND THE BOHEMIAN MASSES)

By Robert Schwinner

Gerlands Beitrage zur Geophysik,
Band 23, Heft 1, 1929, pp. 35-92.

Geologic-petrographic considerations lead to the theory that an old mountain-system (trending from north to south) is hidden under the Calcareous Alps (striking from east to west) and makes a connection between the Bohemian mass and the Central Alps. The examination of the measurements of gravity gave in the first place the startling fact that the idea furnished from the doctrine of isostacy (particularly in the version of Airy) of the deeper part of the crust of the earth must be essentially modified with regard to the Alps. That the 50-kilometer thick flakes of the crust consisting of gneiss (Sial) are dipping more or less, according to the load of the mountain formation in the heavy basis magma (Sima) is most sufficient to explain the observed anomalies of gravity; the high gradients demand flatter dispcsed disturbing masses, and these may be suggested, according to geological and geophysical data, as branches of the subjacent heavy "Sima" rising up nearly to the surface through the fissures between the blocks of "Sial."

Such fissures correspond to the principal dislocations of the mountain ranges of this age (border of the Alps, the zone of graywacke of Upper Styria). But even the corresponding structure and distribution of masses of the old mountain chains denuded at the surface and entirely masked by younger covering mountains are preserved in the substratum.

The crossing of the trend lines of the two mountain systems of different age, which can be read in the isograms, confirms the geologic thesis mentioned at the top. Difference in material involves different elastic reaction. Therefore, at earthquakes which are frequent and known on account of their paradoxial extension, these fissures, and indeed those lines which are geologically postulated and known by the disturbance of gravity, can lead, as leading plains of "conducted waves" (Uller), the seismic energy as "transversal quakes" under the Calcareous Alps to the Bohemian mountain mass; or, on the other hand, can stop their spreading by acting as "seismic bars" in the strike of the Alps.--Author's abstract.

GEOPHYSICS

By G. Angenheister

Terrestrial Magnetism,
vol. 34, No. 1, March, 1929, p. 62.

H. D. Harradon gives a brief review on the first part of the twenty-fifth volume of Wein and Harms, Handbuch der Experimentalphysik, which bears the general title of "Geophysik." The authorship of this work has been shared by six prominent specialists working under the general editorship of G. Angenheister.

"The work is divided into seven main sections. In the opening section, A. Defant gives a general discussion of the physics of the atmosphere, considered under the two headings of Statics and Dynamics. The two following sections on tidal vibrations of the atmosphere and atmospheric optics are by J. Bartels and W. Milch, respectively. The fourth section contains a general discussion of atmospheric electricity by H. Benndorf. Next follows a treatment of northern lights by L. Vegard; he describes the auroral phenomena with reference to form and altitude, relationship with terrestrial magnetism and sunspots, the various theories regarding polar lights and the physical investigations connected therewith, the spectrum of the aurora, and the conditions of the upper layer of the atmosphere, and concludes with an account of an experimental test of his own hypothesis. K. Büttner contributes a section on the penetrating radiation. G. Angenheister and J. Bartels conclude the volume with an essay on the earth's magnetic field, treating the subject under the following headings: Instruments and methods of measurements; the permanent field and secular variation; periodical variations and magnetic activity; magnetic disturbances. The volume is provided with a convenient subject and author index."

The German title of the book is as follows:

Angenheister, G. - Geophysik. I-Teil, unter der Redaktion von G. Angenheister, J. Bartels, H. Benndorf, K. Büttner, A. Defant, W. Milch, L. Vegard. Wien-Harms, Handbuch der Experimentalphysik, Band 25, I Teil, Leipzig. Akademische Verlagsgesellschaft m.b.H., 1928 (XIV + 699 mit 185 Abbildungen).--W. Ayvazaglou.

POSSIBILITIES FOR GEOPHYSICAL PROSPECTING OF PLACERS

(In Russian)

By K. P. Kosin

Gorny Journal, Moscow, No. 4, April, 1929, pp. 548-553.

So far very little attention has been paid to the possibility of geophysical prospecting of placers.

Referring to two articles which appeared in the Engineering and Mining Journal Press (A. Gibson, Magnetometric Determinations Applied to

Placer Mining," vol. 144, No. 25, 1922, and K. Laylander, Magnetometric Surveying as an Aid in Exploring Placer Ground, vol. 121, No. 8, 1926), Kosin draws the conclusion, based on the results of investigations obtained by Gibson and Laylander, that geophysical methods, especially the magnetometric, can and should be used in certain cases. Of course some favorable conditions are necessary. Therefore, geophysical prospecting must be accompanied by drilling and digging, as the latter work may serve not only for the verification of the results but also as material for the interpretation of them.

According to the author, successful results may be obtained--for example, when a placer deposit already contains 0.5 per cent of magnetite.

No geophysical prospecting of placers has been tried so far in the U.S.S.R.

In the last part of the article Kosin discusses theoretically the possibility of using electrical and especially seismic methods for prospecting placer deposits. He concludes that in order to enlarge the sphere of geophysical prospecting for placer deposits a detailed examination of technical possibilities is desirable.--W. Ayvazaglou.

URGES LABORATORY DEEP IN THE EARTH

By Prof. H. Shapley

The New York Times, Nov. 27, 1929, p. 14.

This article is an address delivered by Prof. Harlow Shapley, director of the Harvard Observatory, before the American Geographical Society in New York.

Speaking on what he termed "the third dimension in geography," Shapley asserted that failure to study the interior of the earth beyond a small depth close to the surface constitutes a very serious hiatus in scientific inquiry. For example, by boring deep into the earth it might prove possible to tap the earth's internal source of heat for utilitarian purposes; the knowledge of the mineral wealth of the region bored would be important besides. Therefore, Shapley urged the establishment of a series of laboratories, called by him "Plutonic laboratories," at various depths along a vertical shaft. The installation of such laboratories, although a scientific adventure of great difficulty and expense, would mark an important epoch in man's attempt to understand the earth.

These laboratories may contribute to the knowledge of many important fields in geophysics and astronomy, such as: Investigations of the irregularities in the length of the day and the pulsation of the earth's crust; the thermogradient of the earth and its bearing on the origin of the earth and its source of heat; the nature of the radio activity in the rocks and its variation with depth; studies of earthquake waves; experiments on relativity--for example, the problem of the "ether-drag;" and the problem of the floating of the continents--a test of phases of the Taylor-Wegener hypothesis which holds that America is drifting away from the African and Eurasian continents.

Many smaller scientific problems, such as experiments on living organisms in situations free from cosmic radiation, study of gravity variation and the lunar tides in the body of the earth, as well as the dating of the ice ages and the interglacial periods could be studied also.

Part of Prof. Shapley's address was devoted to a discussion of the earth-moon system.

Pointing out that both theory and observation indicate that the moon has been gradually receding, this recession being about 7 feet per century, Prof. Shapley declared: "The future of the earth-moon pair seems to be that the day will grow gradually longer, and also the month, until they are the same, at about 47 of our present days. This will require, according to Jeffreys, some 50,000,000,000 years. After this, the moon will again approach the earth gradually until it is broken up by tidal disruption just before it gets to the earth's surface. This remote catastrophe may result in a ring of fragments circulating around the earth much like the ring around Saturn."--W. Ayvazoglou.

A BRIEF REVIEW OF ARTICLES APPEARING IN THE TWO FIRST NUMBERS OF THE "BOLETIN DE LA ASOCIACION GEOFISICA DE MEXICO."

Vol. 1, Nos. 1 and 2, July and August, 1929, pp. 5-8, 33-52.

The bulletin issued recently by the Geophysical Association in Mexico, the two first numbers of which (July and August, 1929) have just appeared, contains several articles dealing with geophysics and their application in prospecting for minerals and oil.

The following articles may be mentioned:

1. Importance of Studying Geophysics (Importancia del estudio de la geofisica) (No. 1), by Eng. R. Monges L. The author gives a brief description of the development and study of geophysical methods of prospecting in different countries.

2. Geophysics as Applied to Minerals and Oil (La Geofisica aplicada a la minera y al petroleo) (No. 2), By R. Monges L. This is a brief description of different geophysical methods of prospecting (gravimetric, seismic, electrical, magnetic, radioactive, acoustical, and thermic). Preferences for application of one or another method in the search for minerals and oil are discussed, and the connection of geophysics with geology is mentioned.

3. Physical Principles of the Gravimetric Method of Prospecting, by P. Nikiforov (see Geophys. Abs. 2). This is a translation of Nikiforov's article from Russian into Spanish made by Prof. Eng. Juan Korzujin. (The article is to be continued.)

4. Application of Geophysical Methods of Prospecting in Practice (El Empleo de metodos geofisicos en la practice), by Eng. Kurt Ritzau.

In the part of the article given in No. 2 of the Boletin, Ritzau discusses questions concerning the objects of application of geophysics and the fundamental principles of each method of geophysical prospecting. The article is to be continued.

A list of members of the recently created Geophysical Association of Mexico is given in issue No. 2 of the bulletin.--W. Ayvazoglou.

SCIENTIFIC IMPORTANCE OF APPLIED GEOPHYSICS

(In Russian)

By V. A. Gorsky

Transactions of the Fourth Meeting of Russian Academic Organizations Abroad, 1929, pp. 85-91.

In the article Gorsky expresses his opinion on the desirability of unifying the methods used for studying the earth's crust with the new means offered by geophysics. Taking into consideration that (1) the natural constant static fields of force and conditions of the geoid as a whole (gravitation, magnetism, electricity, heat and radioactivity) and (2) the artificial, changeable dynamic fields of force (electromagnetic and elastic), are known theoretically under the condition that the geoid is a homogeneous body, determination of the geological reasons for the deformations of the normal fields caused by the heterogeneity of the earth's crust can be made by measuring at different points of the surface of the earth the values characterizing these fields.

After briefly stating how these geophysical means may be applied, Gorsky concludes that with the aid of systematic geophysical researches on a large scale, the structure of the geoid, laws of the formations, and the complicated relations of different factors in the process of the movement and life of the earth's crust can be studied. The author proposes to unify this study into a special science--"lithospherology."

Being closely connected with the problems and sphere of work of geophysics, geology, and geochemistry, lithospherology may unite all these separate sciences into a definite and harmonious systematic study of the complicated relations existing in the earth's crust.--W. Ayvazoglou.

8. GEOLOGY

DECOUVERTE DE PETROLE DANS LA REGION DE L'OURAL

(DISCOVERY OF OIL IN THE REGION OF THE URAL)

Editorial Note

La revue petrolifiere, Paris,
Nos. 324 and 329, 1929, pp. 779 and 961.

The discovery of extremely vast oil deposits in the region of the Ural near Tchussovskie Gorodki, about 50 kilometers from the city of Perm, is

reported. The importance of this discovery is very great not only on account of the richness of oil (estimated by Prof. Strijev at about 300 millions of tons) but also because of its geographical location in the center of a highly developed industrial, especially metallurgical, region. The ability to transport the oil to all places of the U.S.S.R. by using the big rivers Tchussovaya, Kama, and Volga is a very favorable feature besides. Two and one-half millions of roubles have already been appropriated by the government for the accomplishment of detailed investigations.

The data obtained by the analyses of different samples of oil discovered recently are as follows:

	Per cent
Gasoline - - - -	20
Kerosene - - - -	12 - 15
Oil-gas - - - - -	12 - 20

The remainder is bituminous residue.

The gasoline is rich in aromatic hydrocarbons, the characteristic properties of which are on initial boiling point of 66° and of final point of 175°; 31 per cent evaporates at 100° and 93 per cent at 160°.--W. Ayvazoglou.

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DEPARTMENT OF COMMERCE
UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

REDUCING ACCIDENTS FROM FALLS OF ROOF IN COAL MINES

PART I - SIX ESSENTIALS FOR MINE ROOF SUPPORT
BY - J. W. PAUL

PART II - METHODS AND IMPORTANCE OF ROOF TESTING
BY - H. TOMLINSON

PART III - ACCIDENTS IN COAL MINES DUE TO FALLS OF ROOF
BY - C. W. OWINGS

I. C. 6225.
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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

REDUCING ACCIDENTS FROM FALLS OF ROOF IN COAL MINES[1]

Part I. Six Essentials for Mine Roof Support

By J. W. Paul[2]

INTRODUCTION

The material that composes the roof in coal mines differs in strength according to variations in its physical nature and characteristics. Some material such as limestone and massive sandstone (free from slips) and carbonaceous shale or bone, will often remain in place in narrow work without the use of timber or other means of support, but in wide places the use of supports becomes a real necessity. Roof material composed of shale, fire clay or small laminations of sandstone, or shale or sandstone interspersed with laminations of thin coal, presents conditions that call for timber support in both narrow and wide work. However, in narrow work such unstable roof material may be taken down until the strata assume an arch that is self-supporting for overlying firm strata such as a roof coal, a tough weather-resisting shale, sandstone, or limestone.

The proper support of the roof in roadways used for haulage or travel adds to the economy of operation by preventing delays in transportation and increases the safety of those who travel these roadways.

Roof material that is prone to fall should either be taken down promptly or immediately given support by adequate timbering.

The owner or operator of mines is presumed to furnish the timber, cap pieces and wedges that are necessary for use and to be placed near the point where they are to be used. (This is essentially the requirement of all State mining laws.)

The general practice in bituminous mines is for the operator to employ timbermen to look after the support of the roof along all haulage and traveling roads; in the rooms and at faces of entries the miner is assigned the duty of setting the props. In modified methods of mining and in concentrated practice, the timbering in all parts of the mine is frequently done by special timbermen or crews.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6225."

2 - Senior mining engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

It has been found that in certain coal mines little attention has been given to regulations for timbering; in others the matter has been fully considered; observations made at the latter mines have prompted the presentation of the following six essentials for mine-roof support in the interest of the former. In any scheme for protection against roof hazards there must be:

1. Some definite system of setting timber;
2. Diagrams showing how and where to set the timber;
3. Instructions given the miners;
4. Supervision and regular inspection;
5. A record kept of any neglect to set timber; and,
6. A penalty for neglect to observe the timber regulations.

1. System of timbering. The system adopted must be chosen with reference to the need for supporting the roof; the need for this support is determined by a study of the roof material and its tendency to fall if not supported by timber. In some mines it is found that no timber is required on narrow entries because the roof remains sound without timber supports; at places where it is unsound the loose material is taken down or such timber support is placed to make it secure as is deemed necessary by an experienced mine foreman or timber boss. There are some mine roofs which will not remain in place without timber or other means of support. In all mines some definite method of roof support should be adopted, after an intensive study of the roof conditions has been made; this is a duty that devolves on the underground manager or superintendent. The method adopted should have the approval of the State mine inspector. There is a certain economic pressure in mining that calls for an unobstructed right-of-way along haulage roads to insure no delay in transportation.

2. Diagrams showing where to set timber. A study of the roof and the way in which it is influenced by the mining method used will enable a practical mine official to determine what should be the minimum requirements in the setting of timber, the distance between props, the number of rows of props, and whether crossbars are needed. In the immediate vicinity of the place where a miner is at work he should be protected from falls of the roof by posts set against the roof, the number depending on the character of the roof.

When the minimum requirements have been decided upon, a diagram showing the relative positions of the posts and the working face should be prepared; this should be placed on a bulletin board near the entrance to the mine in plain view of the miners, also at various places within the mine.

3. Instruction of miners. Each miner should be personally instructed by one of the underground officials in the setting of props for support of the roof in accordance with the regulations adopted; the miner should be cautioned to add additional posts when in doubt about the safety of the roof, and in all cases to take down, by the use of a bar, all loose roof material. The men at the face should be instructed in the method of testing the roof by the use of iron bars and, at the same time, holding the fingers of a bare hand against the roof being tested. This is known as the vibration method, and is more to be depended upon than sound alone.

4. Supervision and inspection. The greatest guarantee against the probability of accidents by falls of roof or coal is efficient supervision and inspection by a competent underground official who believes in strict adherence to the regulations adopted for the support of the roof. The supervising official must make his visits to each man at such intervals as will assure him that the work is being conducted in accordance with the regulations and with such verbal instructions as the official may have given. Where a workman is found under dangerous roof the official should remain to see the place made safe, or he should withdraw the man from the place.

5. Record of neglect. There is no more effective practice than that of keeping a daily record of the inspections made of each working place and recording any neglect on the part of the workman to support the roof properly. In this manner the indifferent or careless workman may be identified, and can be given special instructions and supervisions if he is retained in his employment.

6. Penalty for neglect. Where timber regulations are neglected, the careless workman is liable to be injured or killed, and the operator must defray the cost of compensation as well as incur the stigma of being charged with the loss of a life. There should be some schedule of penalties for those who are found to be negligent of their own safety in failing to observe or comply with the timber regulations; these penalties should apply not only to the workman but also to the supervising official who condones violations of the regulations.

SUMMARY

1. In any coal mine there should be some definite system for setting timber, since such a system is in the interest of economy in operation and of safety to the workmen.

2. The adoption of diagrams for the use of underground officials and miners is a great aid in promoting regularity in setting timber.

3. With a definite system and diagrams illustrating the manner of placing timber, the instructions of miners by the underground officials is more easily accomplished, and uniform practice is made possible.

4. Rigid inspection and close supervision are most essential to establishing methods which have to do with support of mine roof.

5. The use of timber or other forms of roof support if not properly set may be the cause of roof failure and a record should be kept as to the manner in which timber regulations are executed or neglected.

6. Without a plan or a scheme for the accomplishment of a definite course of action, there can be little or no discipline; and to enforce discipline there must be some penalty for neglect to follow instructions as set forth in the plan or scheme - in this case systematic timbering.

RECOMMENDATION

Where a plan of action has been adopted for roof support as a means of protection against accidents from falls of roof, it is recommended that a trained and experienced section or face boss be employed for each section of the mine to observe the roof constantly, to persistently point out to the miner, in his working place, all roof dangers, to give instructions as to how to safeguard the men from the dangers, and to see that his instructions are promptly carried out.

Part II. Methods and Importance of Roof Testing.

By H. Tomlinson[3]

INTRODUCTION

The concluding sentence of reports on mine accidents, especially those from falls of roof and coal, frequently reads "this accident is classed as unavoidable." In reality, it could probably be shown that if the roof had been tested by a reliable method the loose rock or coal would have been detected; and if the loose material had been taken down or securely timbered, a life probably would have been saved. If the same time were spent in planning and rigidly enforcing rules and methods of roof testing and care of roof as is now spent in seeking to place the responsibility on the victim when an accident occurs, the cause of the accident in a majority of cases now reported would be eliminated and many lives and much money paid on compensation claims would be saved.

TESTING ROOF

To test the roof properly in a working place comes as natural to a good miner as it does to the driver of an automobile to place his foot on the accelerator; to handle his automobile safely the driver had first to receive instructions from a competent driver. This is also true of a competent coal miner, who rarely gets injured from a fall of roof or coal. However, there are many coal miners in the coal mines today who have not received any training in how to care for the roof or how to test it properly.

This fact is brought to mind clearly in considering European mines, where the longwall method of mining, with its roof control problems, is used. The roof must be tested every few minutes during the shift, and yet, in comparison with the number of men employed at the working faces, the accident records from falls of roof and coal are much lower than ours.

SOUNDING THE ROOF

After observing thousands of coal miners at work the opinion of the writer of this circular is that many underground workers do not know how to test the roof to determine its safe or unsafe condition. The general method of testing

3 - Associate mining engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

the roof is to "sound" it by striking it with a bar, either wooden or iron or perhaps with an iron tip, and with a miner's pick, although one large coal mining company specifies that all roof testing be done with a steel bar of definite minimum weight without weld or joint. The character of the sound is an indication of the stability of the roof. If the sound is hollow or "drummy" the roof is probably loose and dangerous; it should then be taken down or supported with timbers. If a drummy sound is not noted that does not indicate that the roof is safe, for if a dangerous piece of roof is large, it frequently does not give off hollow sounds that may be detected by the ear.

VIBRATION TEST

Inasmuch as the character of the roof can not always be determined by sounding, a more reliable method should be used. The vibration method, which has been found to be more dependable than sounding and is applicable to coal beds less than 8 feet thick, consists of placing the bare fingers of one hand lightly against the roof and with the free hand striking the roof sharply but lightly with an iron or steel bar. The vibration of the roof against the fingers indicates the condition of the roof. In coal beds 8 to 12 feet thick, a rod held in the bare hand against the roof may be used to detect the vibrations.

SOME ROOFS DO NOT GIVE INDICATIONS

Undoubtedly there are certain types of roof whose soundness or looseness can not be determined by sounding or by the vibration method of testing. The roof may have slip planes or cracks, "Kettle bottoms" or fossil stumps, inverted "horsebacks," and clay veins. The presence of slip planes and fossil stumps may be hidden by a thin layer of coal. Testing the roof or coal may fail to reveal the presence of these slips, cracks, and unusual conditions, so that even testing is not always effective. Wherever the condition of the roof is doubtful, systematic timbering should be used. If a coal bed or a section of a bed is known to have unsafe roof conditions, a definite system of timbering should be used in addition to testing the roof. From 85 to 90 per cent of roof-fall accidents occur at or near working faces; however, the other 10 or 15 per cent of the accidents occur along haulage roads, air courses, and similar passageways. Therefore, it is necessary to make roof testing sufficiently extensive to include all open sections of the mine through which men may be required to pass. This phase of testing is a part of the routine work of mine officials and as such is included under supervision.

The vibration method of testing roof should be employed, and rules should be adopted by all operating companies to provide that the roof and sides of working places shall be tested by the workmen when they go on shift, immediately after firing any shots, when a supervising official makes his visits, and at intervals not exceeding two hours during the shift.

The adoption of this rule, and its thorough enforcement would be a great factor in preventing accidents from falls of roof and coal.

SUPERVISION

Adequate supervision is important in preventing accidents from falls of roof and coal and in the economical operation of a mine.

The mine should be so divided into districts that a supervising official can visit each working face three to four times during the shift, in order to properly supervise and instruct the men in his district in their work.

Many of the workmen who become miners are "wedded to the mine" after a few months experience and like their work and surrounding conditions; but there are others who enter upon mine work because no other employment is available and continue in the mine only until they find employment in other industries that suit their choice. The workman who adopts the mine for his life work is amenable to discipline and may be taught to exercise proper care for his safety, whereas the workman who takes a miner's job as temporary employment until something more attractive "turns up," is not vitally concerned with learning the tricks of the trade and becomes a liability to his employer in the matter of an accident-prevention program. Many experienced miners become hardened to the hazards surrounding their work and often take chances that offer a serious risk, and the inexperienced often take chances without realizing the seriousness of the risk; therefore, with this condition in mind, it is essential that there be efficient supervision to insure that the workmen follow instructions and regulations for their protection against accidents from falls of roof.

Each supervising official should be required to carry a tool for the purpose of testing the roof in each working place visited. He should also know definitely what to examine in each place, such as the roof, posting, standing coal, manner of securing the car, placement of shots, and quality of production.

A point in the mining law of many coal mining States that is usually overlooked is that the mine foreman or any supervisory assistant of the foreman is responsible for the safe condition of the roof on entries and the roadway in rooms as well as at the working face. It is indeed rare to observe a mine official testing the roof except at or near the working face. All parts of the mine traveled by men should have the roof tested every day and in the places where men are working the roof should be tested at least every two hours.

An important provision of some mining laws is the requirement that a mine foreman who discovers a dangerous condition shall remain at such point of danger until the place is made safe or shall require the workman to vacate the place until it is made safe. This requirement is an integral part of the practice of testing the roof and coal to determine the danger from probable falls and to make the place safe by taking down the dangerous material or promptly timbering it.

SYSTEMATIC METHOD OF ROOF SUPPORT

A definite method of roof support should be adopted that will fit the condition of the individual mine and provide a maximum distance between supports. Where a systematic method is used it is only necessary for the miner to rely on his own judgment when special conditions arise.

A large-sized cap piece placed at right angles with the roof slips should be used on all posts.

A systematic method of applying sprags or leaners should be adopted for all overhanging coal while the faces are being cut by mining machines.

The following instructions for testing roof, as issued by the general manager of a large coal mining company, are concise and contain much valuable information.

TO ALL COLLIERY SUPERINTENDENTS:

Please impress on all your underground officials the necessity of their seeing that these instructions are carried out and that the men under them are provided with necessary equipment.

Lay particular stress on the necessity of their explaining to the miners why we want each working place equipped with a special implement for the testing of roof. The reason is that having such a tool the miner will not be tempted to use an improper apparatus such as a copper-headed drill or a part of his drilling machine with mechanical attachments or a bar of steel too light in weight.

HOW TO TEST THE ROOF

The test shall be made by the miner standing in such a position that he can place the fingers of one hand on the roof so as to feel the vibration caused by lightly bumping the roof rock with a steel bar. This bar shall be heavy enough to give a decided "thump" but not so heavy as to be unwieldy. Its length will be regulated by the thickness of the bed, as mentioned later.

WHEN TO TEST THE ROOF

Each miner shall test the roof of his working place immediately he enters it to begin his shift and shall test again after the firing of each blast and as often in addition as is necessary to make sure of the safety of the working place.

EQUIPMENT FOR TESTING

Each miner should provide himself for testing purposes, with a solid steel bar. We insist upon a solid bar, without attachments or welds, in order to guard absolutely against misinterpretation of sound due to vibration of the bar itself.

INTERPRETATION OF TEST

Each miner must be instructed by the sectional foreman as to the necessity of using while testing the roof, not only his senses of sight and sound but the sense of touch. A test of each individual must be carried out by this sectional foreman to determine whether that individual properly interprets the vibrations which are received through the fingers when a piece of loose roof is lightly rapped. This test must also include a determination as to whether each individual properly interprets the different sounds which are given off by loose roof and by solid roof when tapped.

WHAT TO DO AFTER TESTING

Each individual must be made to understand that when the test indicates that the roof is of questionable character, he must make the place safe,

preferably by taking down the loose rock. If this is not possible he may properly support the loose rock with props or cross timbers.

TESTING ROOF OVER
12 FEET IN HEIGHT

The miner must be provided with a horse or horses of sufficient height and plank of sufficient length so that he can erect a scaffold from which to get into the proper position to test the roof in accordance with the above instructions. In exceptionally high places special arrangements will have to be made for erecting scaffolds and for cooperation between miners in adjacent chambers when roof is to be tested. The miner should provide himself with a solid steel testing bar weighing not less than 15 pounds (1" diameter by 6' long weighs 15 3/4 pounds).

TESTING ROOF 7 TO
12 FEET IN HEIGHT

The miner must be provided with a small horse or horses and plank so as to be able to get into a position that will enable him to test the roof in accordance with the above instructions. He should be provided with a solid steel testing bar weighing not less than 15 pounds (1" diameter by 6' long weighs 15 3/4 pounds).

TESTING ROOF UNDER
7 FEET IN HEIGHT

The miner should be provided with a solid steel testing bar weighing not less than 12 pounds. The length of this bar will vary with the thickness of the seam. It should be about six inches shorter than the thickness of the seam.

The following sizes are suitable for the bed thicknesses noted:

	Diameter	Length	Weight	Price
6½ ft. and over	1"	6'	15½ lb.	$0.99
5½ ft. to 6½ ft.	1"	5'	13 "	0.65
4½ ft. to 5½ ft.	1"	4'	10 "	0.40
3½ ft. to 4½ ft.	1¼"	3'	12½ "	0.50
2½ ft. to 3½ ft.	1¼"	2'	12 "	0.48
Under 2 ft.	1½"	18"	9 "	0.36

TESTING ROOF WITH HAMMER

This practice is dangerous if the man stands immediately beneath the roof he is testing, but is sometimes advantageous in detecting a large bell or slab. It is therefore to be discouraged unless the man using this method shows by his actions that he is aware of the possible danger. If any individual prefers to use this method the sectional foreman must make sure that the miner keeps out from under the piece he is testing. The foreman should endeavor to persuade the miner who uses the hammer method to also provide himself with the proper size of testing bar.

Part III. Accidents in Coal Mines Due to Falls of Roof

By C. W. Owings[4]

INTRODUCTION

The adage "to be forewarned is to be forearmed" applies directly to the prevention or reduction of fall-of-roof accidents in coal mines; "to be forewarned" may be considered as the result of frequent testing of the roof, and "to be forearmed" may well be taken to mean systematic timbering of the roof in coal mines. A common expression among mining men is "if the roof is loose (Presupposing testing of the roof), take it down; if loose material can not be taken down or if the condition of the roof is doubtful, timber it." While this rule of making the roof condition safe is generally known, testing and timbering of the roof are generally disregarded or imperfectly practiced in many mines, as is indicated by the large number of fatal accidents reported each year from falls of roof.

STATISTICS

During the 10-year period 1919 to 1928, inclusive, 8,909 fatal accidents (or 49.48 per cent of all accidents) in bituminous coal mines of the United States were caused by falls of roof and coal.[5] During the five-year period 1924 to 1928, inclusive, in bituminous coal mines 4,495 fatal accidents of 8,578 underground accidents, or 52.5 per cent, resulted from falls of roof and coal. During the same five-year period, 5,569 of a total of 10,808 underground fatalities, or 52.1 per cent of all underground accidents, were caused by falls of roof and coal in anthracite and bituminous mines.

This record is not one of which to be proud; the fact that nearly three persons are killed by fall of roof and coal for every fatality from underground haulage and that the ratio of fatalities by falls to those from explosions of gas or coal-dust is about 4 to 1, should serve as an indictment against the apparent indifference of many of our mining men toward the taking of definite adequate measures to prevent accidents from falls of roof and coal. It is true that in 1928 there was a decrease in the number killed as compared with the records for 1925, 1926, and 1927, but there has been no appreciable decrease in number of killed during the 10-year period 1919 to 1928, inclusive as shown in Table 1.

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4 - Associate engineer, U. S. Bureau of Mines.

5 - Adams, W. W., Coal Mine Fatalities in the United States; 1927. Bull. 293, Bureau of Mines, 1928, 120 pp. Coal Mine Fatalities No. C. M. F. 84, Coal Mine Fatalities in January, 1929.

Table 1.- Coal-mine fatalities during the years 1919 to 1928 from falls of roof and coal

Year	Number killed	Year	Number killed
1919	1100	1924	1062
1920	1132	1925	1080
1921	1024	1926	1214
1922	905	1927	1149
1923	1167	1928	1064

A study of these figures shows the need of putting forth a systematic effort to reduce and possibly eliminate this major cause of accidents in coal mines. Any concerted effort to eliminate this cause must embrace frequent and effective testing of roof and coal, also systematic timbering supplemented with additional timbering as conditions demand.

SPECIFIC CASES AND REMEDIES

A recent fatality was the result of depending too much on sound to detect loose roof. The coal bed was about 12 feet thick and the roof had a number of slips, some of which were invisible and ran at an an angle of about 45° to the center line of the room. The timber in the place consisted of round posts surmounted by cap pieces 2 by 5 by 20 inches. The posts were 6 feet 4 inches apart, and according to the testimony of a number of mine officials and workmen the place was properly timbered. The pillar was being drawn in this room, and as several falls had been made, there was danger of roof falls, although the roof was considered "good." The fireboss and the safety inspector had each made two inspections of the place, and the assistant foreman had inspected it about an hour before the accident, testing the roof with a wooden tamping bar by the "sound" method. The miner who was killed had also tested the piece of rock about 30 minutes before the accident, also using a tamping bar and sounding the roof. While loading a car a piece of rock 8 by 4 feet by 9 inches fell, inflicting injuries from which the miner died in about three hours.

If the officials had used the sounding rods provided by the company for detection of roof vibration, or if the miner had used the vibration method of testing, the loose condition of the roof might have been detected and the accident averted.

The lesson to be learned from this accident is that the loose character of large pieces of rock or coal can not always be detected, hence the "sounding" method is not always reliable. Another point to be observed is that in places where slips are known to exist in the roof the use of large cap pieces or collars supported by several props is advisable and if pillars are being extracted, posts set 6 feet or more apart do not afford proper protection.

Even where the roof is sounded and found to be unsafe, no benefit will be derived from this safety practice unless adequate precautions are taken to prevent the workman from being caught by the roof if it falls. An account of an accident in a West Virginia mine shows the futility of taking a chance. A timberman had "sounded" the roof and knew it to be unsafe; however, he believed that he would have time to set a crossbar before the slate could fall. While digging

a hole to set the crossbar a loose piece of slate weighing about 75 pounds fell and struck him on the head, killing him instantly. The lessons to be learned from this accident are that certain types of loose roof may be detected by sounding; that when loose roof is detected no one should work under it; and that if the roof is loose enough to fall, it is far better to pull down all loose material rather than attempt to timber it.

Another accident reported to the United States Bureau of Mines indicates the shortcomings of the sound method and illustrates the possibilities of the vibration method for testing coal-mine roof: In a Virginia mine the practice was to lay the haulage track in the center of rooms and set posts on only one side of the track. When the pillars were being drawn, after a mining machine had cut a slab about 50 feet long, the track was shifted to the rib. About 8 o'clock one morning the mine foreman visited a room which was having the first slab undercut. He examined the place, presumably testing the roof by sounding, pronounced it to be safe and continued with his inspection of the mine. About half an hour later a piece of roof 50 by 14 by 2 feet fell from the unsupported roof at the place being undercut. Five men were caught under the heavy piece of rock and killed instantly. While the practice of leaving such a large area of roof unsupported in pillar extraction may have been considered safe as the result of a sound test, nevertheless if the vibration method of testing the roof had been used, the dangerous character of the rock undoubtedly would have been detected. In all events, such a large area should have had some timber support even though the test indicated the roof to be sound.

The value of testing the roof by the vibration method is indicated in the following account of a roof-fall accident:

In a pillar section of a mine operating in the Pittsburgh coal bed about 10 men were congregated at a point about 30 feet outby the breakline of a chain pillar. The general method of testing was to sound the roof, and the official in charge had, so he stated, tested the roof at the start of the shift. About an hour later while these 10 men were talking, one of them placed the fingers of one hand against the roof and then struck the roof a blow with a pick held in his other hand. Immediately he felt a pronounced vibration which startled him and he gave a cry of warning. All of the men jumped, and, with the exception of one man, they escaped; a piece of rock 20 by 6 by 2 feet fell and crushed out the life of one man. It seems probable that if the mine foreman had tested the roof by the vibration method its unstable condition would have been detected and a life would have been saved.

A case showing the responsibility of the mine foreman or his assistant is brought out in the report of a fall-of-roof accident, quoted below:

> At the time of the accident, deceased, a driver, was waiting on miners to finish loading a car of coal when a fall of top coal and rock fell from the roof near the face and struck him, killing him instantly. The accident occurred in a chain pillar between entries and was caused by a large pot rock breaking through the top coal roof which covered the pot rock and made it invisible. The place had been visited by the fore-

man two hours prior to the accident and found safe, although he ordered an extra crossbar put up, as he considered that the distance was too wide between the last crossbar and the face. The instructions were evidently being carried out when the accident occurred. There was no evidence of weight on any of the timbers. The accident was unforeseen and is classed as unavoidable.

The above account clearly shows that the mine foreman betrayed the trust placed in him by not staying until the crossbar was placed. Inasmuch as the accident occurred two hours after instruction had been given to set the crossbar, it is evident that if the mine foreman had remained until the timbering had been finished, the accident would not have occurred. Therefore the accident resulted from failure to observe safe practices, which are: Official remaining until the place is made safe; withdrawal of the workmen; and timbering unsafe roof.

Webster's dictionary defines supervision as the "act of overseeing; inspection"; hence it is evident that a supervisory official should be a man whose experience has trained him to be an instructor and one who observes conditions and gives orders for making the working places safe. If this attitude is observed in connection with preventing falls of roof and coal accidents there will be a great reduction in fatalities and injuries.

I.C.6226.
January, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

AUTOMATIC DERAILING SWITCH AT ROSLYN, WASH.[1]

By S. H. Ash[2] and Rudolf H. Kudlich[3]

The description and attached sketch of an automatic electrically operated derailing switch are based on data furnished through the courtesy of Mr. Thomas Murphy, superintendent of the Northwestern Improvement Co.'s mines in the Roslyn field, Washington, and of Mr. E. M. Brooks, chief electrician of the same company, who designed and installed the apparatus.

The switch is installed on the slope joining the mine and the tipple a short distance above the tipple landing of the Roslyn No. 3 mine of the Northwestern Improvement Co. at Roslyn, Wash., and is for the purpose of derailing runaway trips, and thus preventing injury to the employees on the tipple, or property damage as a result of such occurrences. The device has proved so successful that the company has adopted it as standard equipment and will install it at all of its mines.

The mine cars are handled by rope haulage from the slope to the tipple in 30 car trips; the power is supplied by an electric hoist located underground and at the top of the main hoisting slope. The haulageway from the hoist to the tipple is about 3,000 feet long and consists of about 2,000 feet of rock tunnel with its portal located approximately 1,000 feet from the tipple. The grade to the tipple is from 4 to 8°. In dropping the loaded cars to the tipple, part of the trip occasionally breaks away, and if not derailed in some manner the runaway cars would go through the tipple. Since frequently only a part of the cars in the trip break away, the engineer is not always aware that a runaway has occurred, and manual control of the derailing switch can not be relied upon. With the arrangement herein described the switch is always thrown for the derailment track, and is automatically lined up for the tipple when the trip, if under normal control, reaches a point about 300 feet above the tipple. Should part or all of the trip become detached from the hoisting rope, the runaway trip would reach the derailing switch before it was closed by the automatic device and would be derailed.

The apparatus controlling the derailing switch consists of a controller mounted on

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6226."

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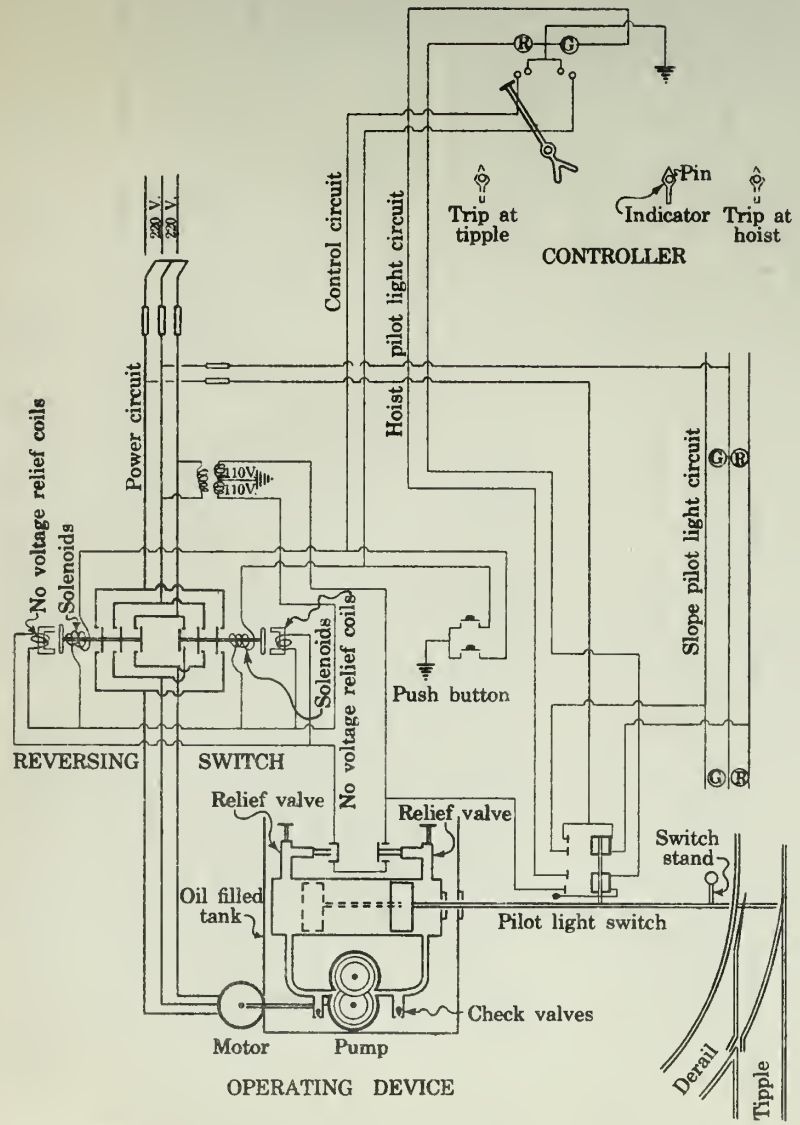


Figure 1. – General diagram of safety device for use where haulage is on slopes

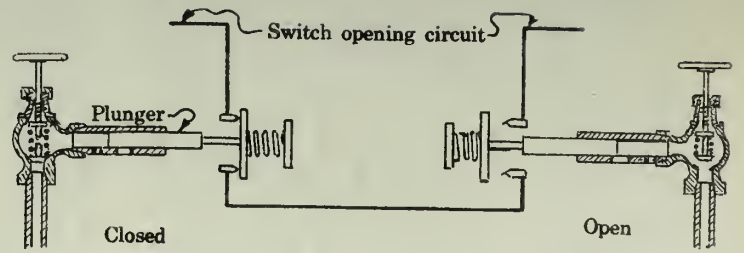


Figure 2. – Detail of relief valves

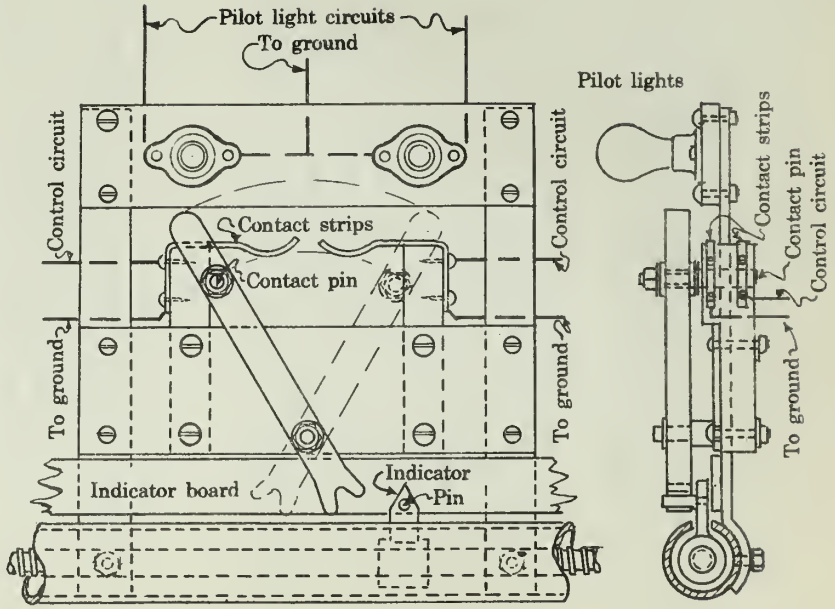


Figure 3. – Detail of controller

the hoist indicator and operated by it; an electrohydraulic operating device at the switch; and pilot lights at the hoist, along the slope, and at the tipple which indicate at all times to the hoist engineer, the rope rider, and the tipple crew the position of the switch. The arrangement and wiring of the apparatus are shown diagrammatically in Figure 1.

The controller (fig. 3) is actuated by the hoist indicator and controls the switch-operating device, causing it to open or close the derailing switch when the trip has reached a certain predetermined point (300 feet above the tipple). It consists essentially of a double throw switch, actuated through a pin and jaw device by the hoist indicator pointer, which makes or breaks the control circuit leading to the electromagnetic reversing switch controlling the hydraulic track-switch operating device. A pin on the hoist indicator pointer engaging the forked end of the contactor arm causes the upper end of the arm to move from left to right or from right to left as the trip approaches or leaves the tipple. A contact pin mounted on the upper end of the arm, but insulated from it, bridges a pair of contact strips and completes one or the other of the control circuits. These circuits actuate the reversing switch.

The reversing switch (fig. 1), mounted near the hydraulic track switch, is a three-pole double throw switch operated by solenoids and having a no-voltage release. The solenoids, connected in the control circuits, throw the switch which governs the direction of rotation of a pump motor which is part of the track-switch operating device. The no-voltage release coils function somewhat like limit switches by being connected through relief valves (fig. 2) on the operating device.

The operating device consists of a piston having its rod connected to the track-switch bridle bar and moved from one end of its cylinder to the other by hydraulic pressure developed by a reversible motor-driven pump in closed circuit with a cylinder having its ports connected one to either end of the cylinder. Pump, cylinder, and piping are immersed in an oil-filled tank, the oil being the hydraulic medium. When the track switch reaches the end of its travel in either the open or closed position or when the mechanism jams, the oil escapes through the relief valve and oil to make up this wastage is drawn in through a check valve on the pump intake.

The relief valves (fig. 2) are spring-loaded to a pressure sufficient to break the track-switch points loose after standing overnight but insufficient to overtax the pump motor or mechanism. (This was found by trial to be an actual thrust of about 300 pounds at the switch points.) When the switch points reach their limit of travel, pressure in the working end of the cylinder increases, the relief valve at that end opens, and the escaping oil forces out a plunger in the discharge pipe. The plunger is held in its normal position by a high spring which exerts just enough pressure to make good contact between an insulated bridge mounted on the projecting end of the plunger and terminal points in the no-voltage release circuit of the reversing switch. A 1/16-inch hole in the wall of the discharge pipe, exposed when the plunger is in the normal position, allows any oil leaking past the relief valve to escape without pushing out the plunger. A second larger hole, exposed only when the plunger is extended, allows the escape of the full discharge of the pump, if the pump has not stopped running by the time the plunger has been extended to the limit.

A push button station located at the switch, having two buttons for "tipple" and "derail" respectively, provides for the working of the switch by means of the operating device, regardless of the automatic control at the hoist. In case either of these buttons

is closed, the indicating lamps and entire mechanism operate as when controlled by the indicator attachment. It is impossible then to operate the switch from any other position until the button control switch is open. By disconnecting the piston rod from the switch bridle, the switch may be thrown by hand. An ordinary weighted, hand lever-switch stand is provided, the lever weight tending to hold the switch points in the normal (derailing) position.

Pilot lights along the slope, at the hoist, and at the tipple are controlled by a switch linked to the track-switch bridle so as to show red when the switch is set for the derail and turn to green as the switch is thrown for the tipple.

Current is taken directly from the 220-volt three-phase power circuit for the track-switch operating device and for the pilot lights along the slope and at the tipple. For the control circuits and pilot light at the hoist, it is stepped down through a 200-watt insulating transformer to 110 to 220 volts with the middle point grounded.

The device operates in the following manner when a trip is being lowered and approaches the derail switch under normal conditions. The indicator pointer moving from right to left (fig. 1) throws the contactor arm over so as to bridge the contacts at the right. This closes the "tipple" control circuit, draws the armature of the reversing switch to the right and starts the motor in the proper direction to pump oil from the left hand end of the cylinder to the right forcing the piston to the left and setting the track for the tipple. When the switch is closed, excess pressure builds up in the cylinder, the relief valve on the loaded end of the cylinder opens, the plunger is forced out and the circuit broken. This causes the no-voltage release coil to become dead and the power switch to open. Meanwhile the pilot light switch is being thrown to the left; closing the circuits on that side and lighting the green pilot lights. As the empty trip is being hoisted from the tipple, the procedure is reversed; the track switch is thrown to "derail" and the red lamps are lighted.

If part or all of the trip should part from the hoisting rope, it would gain speed and reach the derailing switch sooner than the end of the rope being paid out by the hoist. The derailing switch is controlled by the hoist and therefore would not have been thrown to the "tipple" position by the time the runaway trip reached it and would turn the trip aside on to the derail track.

This device was made and installed by colliery electricians from materials readily available or made in the colliery shop. The construction of the controller is seen in Figure 3. The reversing switch is a standard type magnetic three-phase reversing switch with its no-voltage release coils connected in a separate circuit. The pump is an oil circulating pump from a Chevrolet automobile, direct-connected to a General Electric type r.k.t., one-fourth hp., 1,725 r.p.m., 220-volt three-phase motor. The operating cylinder was made from a sleeve bearing casting and is 2½-inch bore by 4½-inch stroke. The relief valves were made from ¼-inch angle globe valves.

There is a definite need for a reliable automatic derailing device wherever slope haulage is used. The device described is worthy of consideration of the officials of every mine in which hauling is on slopes. The adoption of an automatic derailer not only increases the safety of employees, but it also increases the economic operation of slope haulage by decreasing property damage.

10 am

I.C.6227.
January, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

ACTIVITY OF THE HOLMES SAFETY ASSOCIATION IN REDUCING ACCIDENTS IN ALABAMA[1]

By C. E. Saxon[2] and C. W. Owings[3]

The Holmes Safety Association is a national organization interested in safety in the mining, metallurgical, petroleum, and quarrying industries. In 1916 the representatives of 24 leading organizations associated with the mineral industries of the United States met in Washington, D. C., to honor the memory of Dr. Joseph A. Holmes, a man of national prominence and the first Director of the United States Bureau of Mines, an office which he held until his death on July 12, 1915. To carry out the wishes of this friend and benefactor of the employees and operators of the mineral industries and to meet the need for an organized safety movement that would reduce the abnormally high fatality and injury rates then prevailing, it was unanimously decided to establish the Joseph A. Holmes Safety Association as a tribute to his memory.

In 1922 the council directing the affairs of the association thought it advisable to conduct a campaign to establish local chapters. In 1926 the Holmes Safety Association with headquarters at Pittsburgh, Pa., was established and given complete authority to govern itself and to establish State councils and local chapters. Since 1922, 183 chapters have been established, and of this number 43 have been organized in Alabama.

The first chapter of the Holmes Safety Association to be established in Alabama and the second to be organized in the United States was Chapter 2, organized on April 15, 1922, at New Castle, Ala.

On June 10, 1924, the director of the Holmes Safety Association visited Alabama and gave a talk on the Holmes Association work at a meeting of the Alabama coal operators. It was agreed at this meeting that there should be a governing body for all of the Alabama chapters of the association as soon as enough chapters had been organized in the State.

The governing body is called the Alabama Council of the Holmes Safety Association and is composed of representatives of the Alabama Mining Institute, Alabama State Inspection

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6227 "

2 - Foreman miner, U. S. Bureau of Mines, Birmingham Ala.

3 - Associate engineer, U. S. Bureau of Mines.

Department, U. S. Bureau of Mines, insurance carriers, mine superintendents, and safety engineers. The Council meets bimonthly to select subjects for the consideration of the program committee, to appoint committees, to elect officers, and to outline general policies.

The Alabama council gives much of its time and attention to the organization of new chapters and to the maintenance of interest in the less active chapters. Some chapters have been very active and industrious, functioning successfully without the help of the Alabama council; other chapters are unable to hold meetings without outside assistance and call on the council to furnish a speaker for every meeting. A list of capable speakers in and around the Birmingham district who are willing to address the local chapters is kept on file. When a chapter officer calls on the council for assistance, the list is consulted and a speaker sent to the chapter meeting. Some chapters asking for a speaker defray the expenses of the speaker; in some, the council furnishes the speaker with transportation; and in others, the speaker meets his own expenses.

In the hope that the lessons taught may prevent the occurrence of similar fatalities in the future, the Alabama Council of the Holmes Safety Association issues a bulletin each month giving in detail the fatal coal-mine accidents of the previous month. These bulletins are mailed to the operating companies in the district.

The council has sponsored many campaigns on various subjects and has given prizes for the best papers on these subjects submitted by employees of the companies. Below is a list of a few topics that speakers have used at different meetings of the chapters:

Accidents from falls of roof and coal.
Electrical accidents.
Haulage accidents.
Handling explosives.
Unusual accidents.
Necessity for discipline.
Miners' camp sanitation and personal hygiene.
Care of machinery.
Mine gases and ventilation.
How to prevent a mine explosion.
What to do in case of a mine explosion.

At one time 36 active Holmes Safety Chapters in Alabama held regular monthly meetings; at present only 29 chapters are active. Six chapters were disorganized when convict labor in the coal mines was discontinued. Most of the mining communities have both white and negro Holmes chapters but a few companies have only white chapters at their mines. All of the chapters meet at night, after working hours, and generally last for two to three hours. The usual order of business is as follows:

Song or prayer.
Minutes of last meeting.
Solo or quartet.

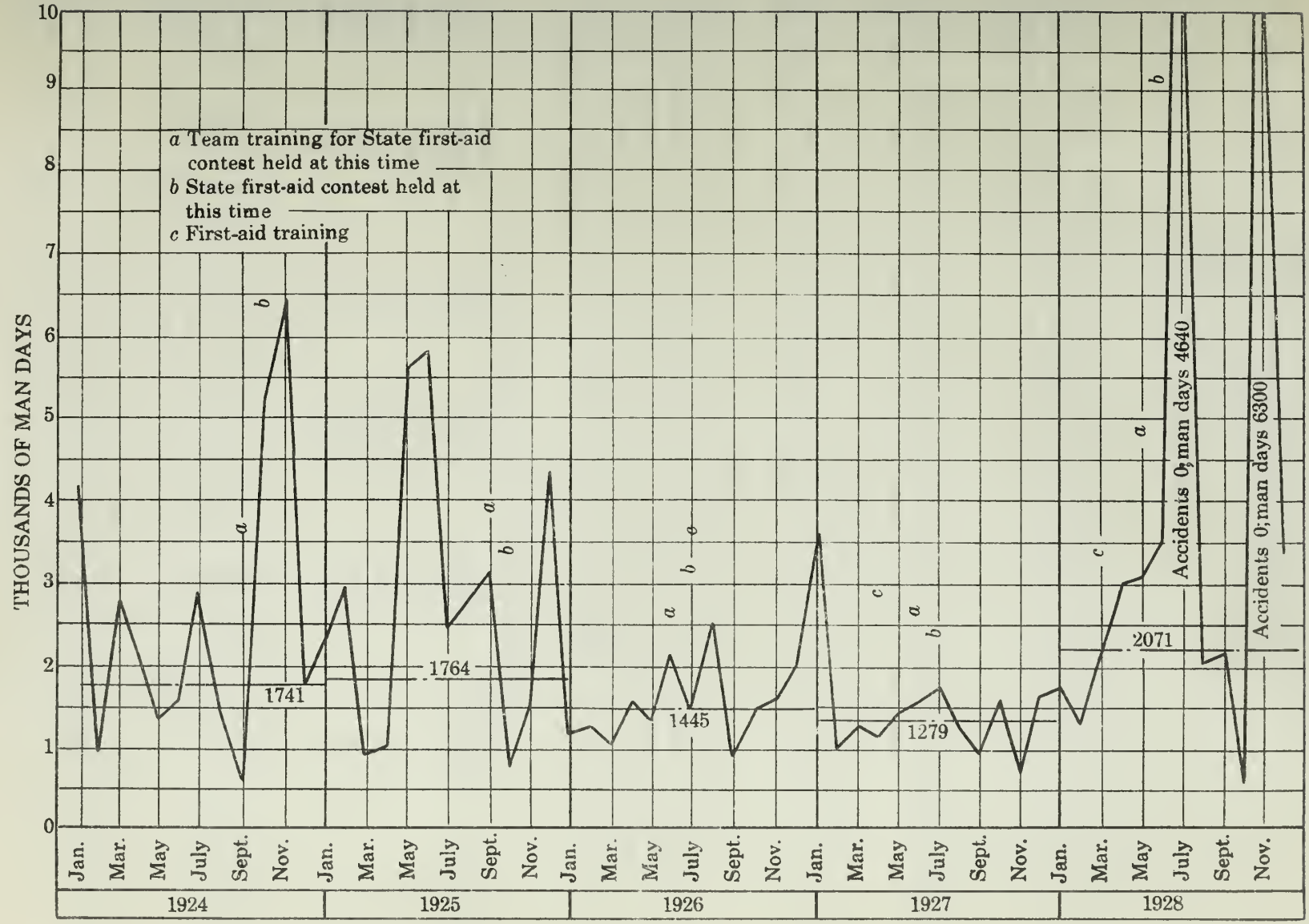


Figure 1. – Man days worked per accident. Company No. 1 (2 mines)

Speaker of the evening; subject, some phase of accident prevention or health work.
Report on accidents for the past month.
Song or recitation.
Refreshments (sometimes).
Adjournment.

During the school term, the chapters frequently enjoy a play or recitation by the school children. Very few of the chapters set aside any time for the practice of first aid, one of the objects for which the chapters were organized. The chapters that have a moving picture machine (and most of them have one) often borrow a picture from the Bureau of Mines or from some other source.

Fees or dues are not assessed except by one or two chapters in the State which require the payment of 25 cents per month for a miner's family. The money collected in this way is spent as the chapter designates, generally for refreshments.

The effect of the activity of the Holmes Safety Association chapters in reducing accidents at mines is shown by the following tables and accident-frequency curves which have been collected from three coal-mining companies that have active Holmes Safety Chapters.

A chapter was organized at the mines of company No. 1 during 1922, but as the company did not keep an accurate record of accidents for some time afterwards, it can not be shown whether or not the organization of the chapter was instrumental in causing a reduction of accidents in 1922 and 1923. However, the company began to keep a record of accidents in 1924, 18 months after establishing the Holmes Safety Chapter. During 1927 one fatal accident occurred for which 6,000 days of lost time are allowed. This accident was caused by a local gas explosion; no one else was injured at the time.

Lost Time Accidents at the mine of Company No. 1

Year	Average No. employees	Days worked	Man days worked	Days lost-time	Number of accidents	Man days per accident
1924	371	216	80,067	448	46	1,741
1925	345	240	82,898	394	47	1,764
1926	377	276	104,022	848	72	1,445
1927	408	207	84,437	6804	66	1,279
1928	335	198	66,278	489	32	2,071

The accident frequency curve (fig. 1) on coal-mining Company No. 1 covering the period 1924 to 1928, inclusive, shows the number of man days worked for each accident with an average figure for each year. There was an accident in 1924 for every 1,741 man days worked; in 1925, for every 1,764 man days; in 1926, for every 1,445 man days; in 1927, for every 1,279 man days; and in 1928, for every 2,071 man days worked.

Coal-mining company No. 2, which operates 10 coal mines began organizing Holmes Safety Chapters at its mines during 1924; white and colored chapters have been organized at each of its mines. The Holmes chapters were organized six months previous to the years involving the accident records below:

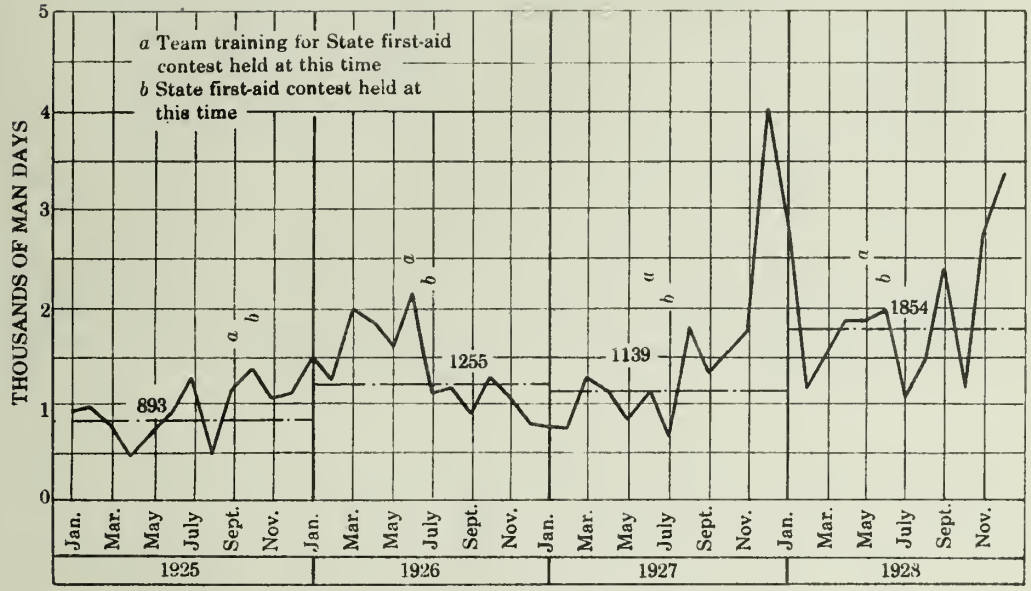


Figure 2. – Man days worked per accident. Company No. 2 (10 mines)

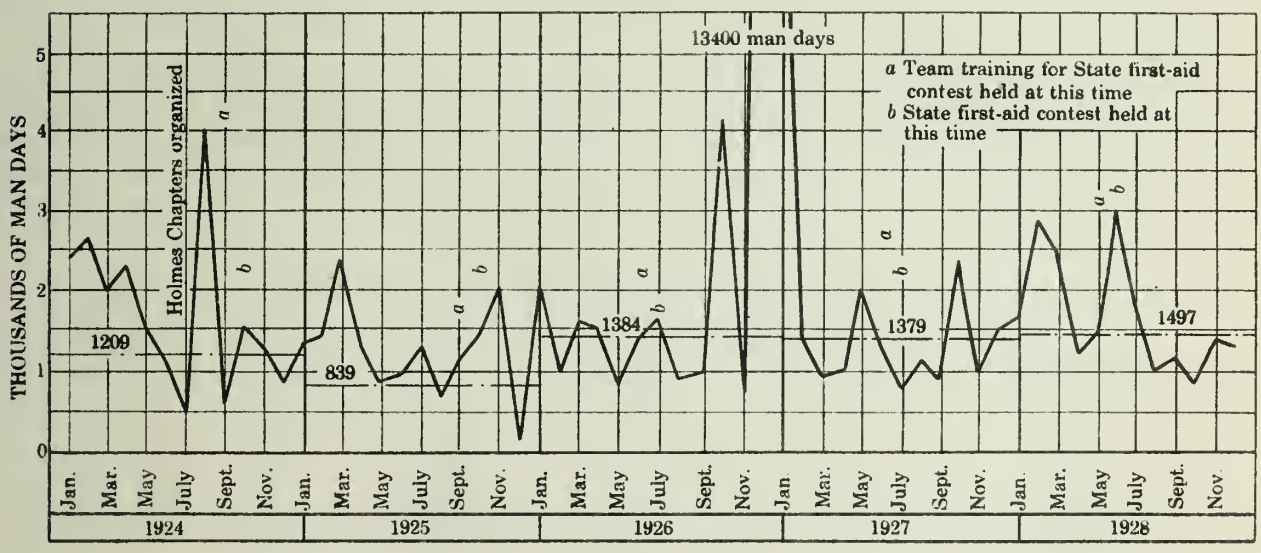


Figure 3. – Man days worked per accident. Company No. 3 (13 mines)

Lost-Time Accidents at company No 2

Year	Average No. employees	Days worked	Man days worked	Days lost	Number of accidents	Number of fatalities	Man days per accident
1925	1821	223	406,153	43,302	455	6	893
1926	1865	248	463,030	43,833	369	6	1,255
1927	1789	199	355,237	31,851	312	4	1,139
1928	1565	179	279,905	25,803	151	3	1,854

Six thousand days of lost time are charged for each fatal accident included in the above table.

The Holmes chapters (both white and colored) of this company are active and meet regularly. This company has started 100 per cent first-aid training; that is, all of the employees at two of the mines have been trained in first aid, and the training of all employees at the other eight mines is being arranged.

The accident-frequency curve (fig. 2) of coal mining company No. 2 from 1925 to 1928, inclusive, shows the number of man days worked for each accident with an average figure for each year. The curve indicates that accidents in this company have been decreasing. During 1925 there was an accident for every 893 man days worked; during the last recorded year - 1928 - there was an accident for every 1854 man days worked, which shows a great improvement.

Coal mining company No. 3 operates mines at three different places, all of which have white and colored chapters of the Holmes Safety Association. Regular meetings are held once a month; attendance is good and the chapters are very active.

The following table gives the average number of men employed, days worked, number of accidents, number of fatalities, and the man hours per accident for the last six months of 1923 and for 1924 to 1928, inclusive. This table is a consolidated yearly report for all of the company's mines. Of the 59 fatalities in 1925, 53 were caused by an explosion in December of that year.

Lost-time accidents at company No 3

Year	Average No. employees	Days worked	Man days worked	Number of accidents	Number of fatalities	Man days per accident
1923	494	116	57,356	41	2	1,526
1924	458	224	102,843	85	4	1,209
1925	531	283	150,266	179	59	839
1926	545	290	158,181	115	4	1,384
1927	572	244	139,364	110	2	1,379
1928	512	246	125,829	84	5	1,497

In the foregoing table "Number of Accidents" comprises all accidents causing 14

days or more of lost time, including fatal accidents. For the number of nonfatal accidents for 1928, subtract 5 (the number of fatal accidents) from 84, the total number of accidents for that year; the number of nonfatal accidents causing more than 14 days lost time for that year is, therefore, 79. This procedure applies also to any other year.

The accident-frequency curve (fig. 3) covering July 1, 1923, to December 31, 1928, inclusive, shows the number of man days worked for each lost-time accident at all mines of company No. 3. The last six months of 1923 show an average of one lost-time accident for every 1,526 man days worked; 1924 shows one accident for 1,204 man days worked; 1925, one accident for every 839 man days worked; 1926, one accident for every 1,384 man days worked; 1927, one accident for every 1,379 man days worked; and 1928, one accident for every 1,497 man days worked, the best annual record since 1923.

The safety records of these three mines do not furnish exaggerated examples, for the work of all the chapters in the State has resulted in a more or less similar reduction of the mine accidents. The effectiveness of the association has resulted chiefly from the support given by all the officials of the State Department of Mines, by the Alabama Mining Institute which is in reality the Mine Operators Association, by practically all of the officials in charge of mines, and by the mine employees. The progress made in permanently reducing the fatality rates is evidence of the advantages derived from safety activities since 1925. The reduction in fatalities is as follows:

Year	Fatalities
1925	162
1926	139
1927	93
1928	67

In the annual report of coal mines in Alabama, C. H. Nesbitt, chief mine inspector at that time, made the following statement:

> The Joseph A. Holmes Safety Association which was formed in Birmingham in 1924 under the auspices of the Alabama Mining Institute, the United States Bureau of Mines, and the Alabama Mine Inspection Department is functioning nicely in its efforts to establish chapters in the different mine villages. They hold regular monthly meetings at which a speaker is provided and safety problems are discussed, having in mind the educating of employees to be more careful in their duties, thereby avoiding accidents.

That the effectiveness of the Holmes Safety Association is fully appreciated by the State Inspection Department is shown by the following excerpt from the 1928 annual report, in which W. B. Hillhouse, the chief mine inspector says:

> The outstanding problem of the coal mining industry of the State is the conservation of life, limb, and property, and to attain

this end the fullest cooperation is necessary from everyone concerned. Alabama is enjoying this cooperation in a marked degree through the activities of the coal operators' Alabama Mining Institute, the Birmingham station of the United States Bureau of Mines, the many Holmes Safety Associations, and the State Mining Inspection Department, and as a result the attainments in the direction of safety for the year 1928 are very marked and gratifying.

The Alabama Mining Institute activities were along educational lines, such as the distribution of pamphlets and circular letters dealing with safety problems. The Holmes Safety Associations, numbering 34 chapters and sponsored by the Alabama Holmes Council, have played an important part in the prevention of accidents by disseminating information concerning their causes and means of prevention. The United States Bureau of Mines is very active in this State in the prevention of accidents by training in first-aid, mine-rescue, and other work. The bureau in some cases makes safety examinations and furnishes to the operating companies reports, including recommendations as to improved safety methods. In doing this it works in the closest possible cooperation with the State Department and all other existing agencies.

W. B. Hillhouse also issued a published statement, dated January 5, 1929, the first paragraph of which reads as follows:

Through the cooperation of the Alabama Mining Institute, the Holmes Safety Association, the United States Bureau of Mines, and the State Mining Department, a forward step was taken in the conservation of human life within the industry. Our record for the year 1927 was not an enviable one, and early in the past year a movement to improve this record was begun, the results of which will surely be gratifying to all who put forth their efforts in behalf of the cause.

The value of the Holmes Safety Association chapters in reducing accidents was well expressed by D. A. Thomas, president of the Montevallo Coal Mining Co. of Birmingham, before the National Coal Association in Cleveland, Ohio, on November 14, 1928. His remarks were, in part, as follows:

Alabama has been gripped a great deal with renewed interest this year in accident prevention and general safety work.

We have 30-odd chapters of this Holmes Safety Association, which, of course, are sponsored by the Alabama Mining Institute and the Alabama Fuel Association. These chapters, which are scattered throughout the State, are actively supported by the management of the different mines. In some cases they have weekly, bimonthly, or monthly meetings.

> This year, at the urgent request of our newly appointed mine inspector, who is most efficient and capable in getting thorough co-operation from all the operators of the State, the officers of the Holmes Association within our State listed all possible speakers throughout the State to visit the different chapters. It resulted in having good speakers from one mine go to another. It got away from that cut-and-dried situation that very often exists in the men hearing the same thing from the same superintendent week in or week out or from local speakers. And as a result of that work, I am glad to say that Alabama has an accident record for 1928 that so far is an outstanding one as compared with 1927.

Cooperation from all agencies concerned is necessary to obtain effective safety organizations. In Alabama the cooperation of the State, the coal-mine operators, and the U. S. Bureau of Mines has made the Holmes Safety Association a real instrument in the reduction of coal-mine accidents. If the mining department of every coal or metal mining State would follow the example of Alabama and cooperate actively with the mine operators and the Bureau of Mines in making the Holmes Safety Association chapters the common medium through which safety may be increased, there is good reason to believe that mine accidents would be greatly reduced.

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FEBRUARY 1930

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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TEACHING SAFETY THROUGH THE APPROVAL PLATE

BY

L. C. ILSLEY

I.C.6229.
February, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

TEACHING SAFETY THROUGH THE APPROVAL PLATE[1]

By L. C. Ilsley[2]

The plate of every approved motored-type machine carries the seal of the United States Bureau of Mines. The heading "CAUTION" thereon calls attention to certain additional information applying to the upkeep of the machine in safe condition.

In the first bureau approval plates, the caution statement was very simple, containing two brief paragraphs with the following wording:

CAUTION

The permissibility of this equipment depends upon the absence of openings in the inclosing casings.

Cover plates should be screwed on tight and the casings frequently inspected for openings.

As the approval work continued the value of the approval plate as a means of assisting maintenance of equipment was recognized, and various items affecting the safety of the equipment as a whole were added until a typical caution statement is now several paragraphs in length. The following example is quoted from the approval plate of a recent permissible outfit:

CAUTION

THIS EQUIPMENT IS NOT PERMISSIBLE EXCEPT WHEN USED UNDER THE FOLLOWING CONDITIONS:

1. GENERAL SAFETY: Frequent inspections must be made to see that all electrical parts, including the trailing cable and the wiring, are kept in a safe condition. There must be no openings into the casings of the motor or other electrical parts. A permissible junction box must be used in connecting to the power circuit unless connection is made in pure intake air or to a permissible power truck. The positive conductor of the trailing cable must be connected to the positive terminal of the control.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6229."

2 Electrical engineer, U. S. Bureau of Mines.

2. FASTENINGS: Cable glands and all bolts, nuts, screws, and other means of fastening must be in place, properly tightened and secured. Motor hand-hole covers must be kept sealed and the control cover padlocked to the casing. Only one sealing tool and key shall be available and they shall be placed in the care of an authorized person.

3. RENEWALS AND REPAIRS: Inspections, repairs, or renewals of electrical parts must not be made unless the trailing cable is entirely disconnected from the circuit furnishing power and the cable must not be connected again until all parts are properly reassembled. Special care must be taken in making renewals or repairs. Leave no parts off. Use new parts exactly like those furnished by the manufacturer. When wiring at the cable entrances is disturbed, the stuffing boxes must be repacked tightly and the gland nuts secured against loosening.

4. CABLE REQUIREMENTS: An approved-type trailing cable protected by a fuse or other automatic circuit-interrupting device must be used. Special care must be taken in handling the cable to guard against mechanical injury and wear. Spliced cables must not be used unless the splices are properly made and vulcanized.

5. 500-VOLT EQUIPMENT: The operating voltage must not exceed 500 volts at the motor terminals.

The Bureau of Mines believes that, of two machines identical in so far as construction is concerned, the one bearing an approval plate will be given not only better but more intelligent attention. The fact that certain definite items are listed in the caution statement and thereby brought to the attention of repairmen or inspectors for their guidance should be decidedly helpful in the maintenance of equipment. When the approval plate is absent there is no substitute for these constructive suggestions, and the repairmen, not being especially trained to look for the little details that may affect the safety of the equipment, will probably overlook some vital maintenance needs. The warning that identical parts must be used in making renewals should also be helpful to the repairmen in obtaining the proper material through the purchasing department rather than some cheaper substitute, as often happens in the best regulated mines. The further warning that no additional parts can be installed without destroying the permissibility of the equipment should be especially valuable in minimizing dangerous, unsafe, and unapproved innovations proposed by an overzealous electrician or mine foreman.

The caution statement on approval plates should be of real service to both company and State inspectors in pointing out essential items for an external check of the condition of machines bearing them. If an inspector is asked what is to be done to maintain equipment properly he has a definite procedure already outlined for him in the caution statement. Any machine maintained in compliance with all the requirements thus outlined will be well on the road to safety, and any electrician that takes such an interest in these requirements as to understand their intent and purpose thoroughly should have no trouble, in keeping his

equipment reasonably safe or at least be able to report upon its degree of safety, as measured by the safety goal that has been set.

External Evidences

Many external evidences come to the Bureau of Mines from time to time indicating that the approval plate is teaching safety lessons in several ways.

To a certain extent manufacturers are put on their honor in placing approval plates on machines. The bureau has no follow-up system in connection with the approval plate similar to the label service of the Underwriters' Laboratories, but leaves the matter of placing the approval plate only on suitably designed machines in the hands of the machine manufacturers. At its Pittsburgh Experiment Station the bureau keeps on file detailed information covering the construction of a given permissible outfit and authorizes the manufacturer to place specifically worded approval plates on such designs as agree exactly with the record drawings. To keep faith with this trust the machine manufacturer does several things that bear directly on safety and aid greatly in maintaining the integrity of approval. One practice of most manufacturers is to have a special factory inspection sheet, which is filled out for each permissible outfit; this record is kept on file by the manufacturers as evidence that when the machine left the factory it was properly designed and assembled. A typical factory inspection sheet for a permissible compartment follows:

Motor for Coal-Cutting Equipment

1. Are there any openings or holes, small or large, in shell other than drawings call for? _____
2. Are all flanges for covers free from blow holes? _____
3. Are all flange surfaces smooth? _____
4. Do all flange joints fit tightly? _____
5. Is width of contact between motor shell and end plate 1-1/4 inches or more; between hand-hole covers and shell 1-1/4 inches or more; between motor shell and pinion end bearing housing 2-1/4 inches or more _____

   (Note: The width of contact and not width of flange is considered by the U. S. Bureau of Mines.)
6. Do holes for top studs in motor shell at gear end cut through? _____
7. Are bottom studs in motor shell at gear end riveted over? _____
8. Are studs in motor shell for cover for brush opening riveted over? _____
9. Do holes for studs in motor shell for end plate cut through? _____
10. Do holes for cap screws in end plate for cap cut through? _____
11. Do holes for studs in motor shell for guard cut through? _____
12. Do holes for cap screws in motor shell for guard cut through? _____
13. Do holes for machine bolts for cable clamp block cut through? _____
14. Do holes for cap screw for cable clamps inside motor shell cut through? _____
15. Are there lock washers under all cap screws on motor end plate, on end plate cover, on guard for terminals? _____
16. Are there lock washers under all study nuts? _____
17. Does the key in the gear end housing have a snug fit in keyway in the motor shell? _____

18. Are proper terminal studs, bushings and collars used through shell? _____
19. Are terminal studs tight? _____
20. Are cable terminals properly locked to terminal studs? _____
21. Is the special hose securely clamped? _____
22. Is the special hose clamp and terminals protected by proper cover? _____
23. Is the key for the gear end housing riveted in place? _____

Because of the special inspection form and the more careful shop work required on permissible equipment factory employees soon recognize that machines that are to carry the Bureau of Mines plate should receive special attention and a certain respect for essential safety requirements is engendered.

Many companies train special service men to look after the maintenance in condition safe for use of permissible machinery in the field. Often these men spend a short time with the bureau witnessing the tests and detailed inspection of equipment and thus become well versed in all the details for which the approval plate stands. These service men are in close touch with practical operation of the machine and have an unusual opportunity to point out the proper way to use permissible equipment and call attention to its misuse. Recently a manufacturer reported that one of these service men, on seeing a machine in a condition that did not comply with specifications on the approval plate, had taken what might be considered a rather high-handed position with the operator; he told the latter that permissible equipment could not be used that way and took the plate off the machine. It is understood that the operator recognized the justice of the service man's position; the machine has since been reconditioned and the plate replaced. This incident shows that to this service man the approval plate stood for something more than a piece of brass.

Some approval plates become broken or lost, and coal companies have been reported to have ordered replacement plates as they would other needed supplies; but owing to responsibility of the machine manufacturer in maintaining the integrity of the approval he must assure himself that a machine is in condition to have the new plate attached. Especially is this desirable since, as already shown, more recent approval plates cover items not considered in the earlier approvals. Some operators have wished to convert flame-proof machines to permissible equipment. This is sometimes done at the mine, but before an approval plate is put on any reconstructed machine the manufacturer must assure himself through his trained service man that the machine is in proper condition to receive the approval plate. Often this requires certain new compartments and complete overhauling of the machine. Certain operators have made use of the service man to give their permissible equipment a thorough inspection which, because of his special training, he is often better qualified to give than a company employee. Thus it can be seen that through the service man and his guardianship of the integrity of the approval plate the operator can not help being impressed with the safety principles for which it stands.

The operator often takes the initiative in carrying out safety principles embodied in the approval plate. One large company has on several occasions refused to let equipment be placed in use, although it was known that the machine was of a permissible design, because the approval plate had not yet been attached to the outfit. This company has some gassy mines, and in these it was desirable that the local officials be accustomed to use no equipment that did not bear the safety label of the bureau. The fact that a number of

the large coal companies are obtaining vulcanizing outfits would seem to indicate that the recommendation that all splices in trailing cables be vulcanized has been taken to heart and recognized as an essential safety precaution. These are encouraging signs that the operator values the safety lessons underlying the caution statement.

State inspectors also value this safety symbol. The Bureau of Mines has received many inquiries from inspectors in regard to the time when the plate could be placed on a given outfit, and in certain instances inspectors did not permit its use until the plate was attached. An inspector feels a security with an outfit that has met all the requirements of the bureau that he can not have with any other equipment, regardless of the safety the manufacturer or operator may claim for it. One State mine department had several of its inspectors spend about a week at the Pittsburgh Experiment Station to familiarize themselves with all the details of approved equipment so that they might better check the maintenance practices in connection with the everyday use of permissible machines.

Where electricity is adjudged to have been the contributing cause of disasters the commision of State inspectors usually recommends the future use of nothing but permissible outfits. The only State issuing approvals has discontinued that practice in favor of approvals issued by the United States Bureau of Mines. The Bureau of Mines has received a great deal of assistance from the approval system in its campaign for safety.

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FEBRUARY 1930

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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LIST OF PERMISSIBLE MINING EQUIPMENT

I. C. 6230.
February, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

LIST OF PERMISSIBLE MINING EQUIPMENT[1]

A complete list of permissible mining equipment, rescue apparatus, and gas masks tested prior to July 1, 1929, was published in Bureau of Mines Information Circular 6172.[2] The present list[3] includes practically all equipment tested and approved by the bureau to and including January 1, 1930. It covers electric air compressors, coal drills, mining machines, loading machines, conveyors, mine pumps, room hoists, mine telephones, rock-dusting machines, switches, electric cap lamps, flame safety lamps, electric hand and trip lamps, flash lamps, methane indicators and detectors, blasting units, storage battery locomotives, tandem locomotives, power trucks, concrete mixers, self-contained oxygen breathing apparatus, and gas masks.

The system under which these devices were tested permits the manufacture after his equipment has passed certain tests prescribed by the Bureau of Mines, to mark his equipment with a seal showing that it has been approved by the bureau. These tests are designed to insure the equipment having the minimum requirements for safety in use. The only object of the bureau in making such tests and publishing lists of permissible equipment is to safeguard the lives of workers and to help lessen the hazards of mining.

PERMISSIBLE MINING MACHINES, COAL DRILLS, ETC.

Approved Under Schedules 2, 2A, and 2B

Air Compressors

1. Type WK-26 compressor; 30-hp. motor, 250-500 volts, D. C. Approvals 117 and 117A, issued to Sullivan Machinery Co., March 12, 1925.

2. Type WK-39 self-propelled compressor; 30-hp. motor, 250-500 volts, D. C. Approvals 120 and 120A, issued to Sullivan Machinery Co., July 28, 1925.

3. Type CP-26G, CP-26D, and CP-26H compressors; 25-hp. motor, 250-500 volts, D. C. Approvals 128 and 128A, issued to General Electric Co., March 21, 1927, and July 16, 1926, respectively.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines, Information Circular 6230."

2 - List of Permissible Mining Equipment, Information Circular 6172, Bureau of Mines, Sept., 1929.

3 - A few equipments which may now be considered obsolete and therefore not obtainable are not included in this list.

4. Type 20 compressor; 15-hp. motor, 250-500 volts, D. C. Approvals 159 and 159A, issued to Ingersoll-Rand Co., Sept. 4, 1928.

5. Type WK-22 compressor; 30-hp. motor, 250-500 volts, D. C. Approvals 160 and 160A, issued to Sullivan Machinery Co., Sept. 10, 1928.

Loading Machines and Conveyors

1. Type 43-A shortwaloader; 50-hp. motor, 250-500 volts D. C. Approvals 122 and 122A, issued to The Jeffrey Mfg. Co., Jan. 8, 1926.

2. Type 44-B conveyor-loader; 50-hp. motor, 250-500 volts, D. C. Approvals 123 and 123A, issued to the Jeffrey Mfg. Co., Jan. 15, 1926.

3. Belt-type conveyor; 5-hp. motor, 250 volts, D. C. Approval 126, issued to Bird Coal Co., June 25, 1926.

4. Shovel-type loading machine; 30-hp. motor, 250-500 volts, D. C. Approvals 127 and 127A, issued to Myers-Whaley Co., July 16, 1926, and Sept. 23, 1927, respectively.

5. Chain-type conveyor; 5-hp. motor, 250 volts, D. C. Approval 129, issued to the Bird Coal Co., July 21, 1926.

6. Type 5-BU loading machine; 25-hp. motor, 230-500 volts, D. C. Approvals 132 and 132A, issued to Joy Mfg. Co., December 29, 1926, and March 22, 1927, respectively.

7. Type 49-A chain-type conveyor; 3-hp. motor, 250-500 volts, D. C. Approvals 133 and 133A, issued to The Jeffrey Mfg. Co., Feb. 10, 1927.

8. Conveyor-type loader; 30-hp. motor, 250-500 volts, D. C. Approvals 135 and 135A, issued to the Sullivan Machinery Co., May 11, 1927.

9. Type 136-EC entryloader; 35-hp. motor, 210-500 volts, D. C. Approvals 138 and 138A, issued to the Goodman Mfg. Co., Aug. 5, 1927.

10. Belt-type conveyor; 1-hp. motor, 220 volts, D. C. Approval 139, issued to the Lorain Steel Co., Aug. 19, 1927.

11. Type A.F.-10 by 8 shaker conveyor; 15-hp. motor, 250-500 volts, D. C. Approvals 149 and 149A, issued to C. H. McCullough Engineering Co., March 29, 1928.

12. Type 48-E power shovel; 15-hp. motor, 210 volts, D. C. Approval 150, issued to Goodman Mfg. Co., May 11, 1928.

13. Chain-type conveyor; 5-hp. motor, 250 volts, D. C. Approval 151, issued to South Fork Foundry & Machine Co., May 19, 1928.

14. Conveyor-type face loader; 25-hp. motor, 250 volts, D. C. Approval 155, issued to Bethlehem Steel Co., August 2, 1928.

15. Shaker-type car filler; 22-hp. motor, 250 volts, D. C. Approval 156, issued to Bethlehem Steel Co., August 2, 1928.

16. Pit car loader; (turntable type) 1-hp. motor, 250-500 volts, D. C. Approvals 166 and 166A, issued to Duncan Foundry and Machine Works, (Inc.), March 13, and July 11, 1929, respectively.

17. Brownie pit car loader; 1 1/2-hp. motor, 250-500 volts D. C. Approvals 167 and 167A, issued to the Brown-Fayro Co., March 27, 1929.

18. Pit car loader; 1-hp. motor, 230-500 volts, D. C. Approvals 168 and 168A issued to the Northern Conveyor and Manufacturing Co., April 5, and Sept. 27, 1929, respectively.

19. Conveyor Sales Co. Shaker conveyor; 10-hp. motor, 250 volts D. C. Approval 171 issued to the Goodman Manufacturing Co., April 17, 1929.

20. Pit car loader; 1 1/2-hp. motor, 250-500 volts D. C. Approvals 173 and 173A issued to the Fairfield Engineering Co., April 30, 1929.

21. Pit car loader; 1-hp. motor, 230 volts, D. C. Approval 174 issued to Bertrand P. Tracy Co., May 22, 1929.

22. Pit car loader; 1-hp. motor, 250-500 volts D. C. Approvals 175 and 175A issued to the Chicago Automatic Conveyor Co., July 26 and June 24, 1929, respectively.

23. Pit car loader; 2-hp. motor, 230 volts, D. C. Approval 178, issued to the Northern Conveyor and Manufacturing Co., Oct. 5, 1929.

24. Pit car loader; 1 1/2-hp. motor, 250-500 volts, D. C. Approvals 179 and 179A, issued to the Mancha Storage Battery Locomotive Co., Nov. 26 and Oct. 19, 1929, respectively.

25. Type L-2 loading machine; 13 motors aggregating 117-hp., 250 volts, D. C. Approval 182, issued to the Oldroyd Machine Co., December 9, 1929.

Coal Drills

1. Type 2-BF drill; 1-hp. motor, 90, 110, and 250 volts, D. C. Approvals 109 and 109A, issued to Chicago Pneumatic Tool Co., Sept. 19, 1922.

2. Type CD drill; 3/4-hp. motor, 110-230 volts, D. C. Approvals 110 and 110A, issued to Martin-Hardsocg Co., Sept. 16, 1922.

3. Type A-5 drill; 3-hp. motor, 110-250 volts, D. C. Approvals 119 and 119A, issued to The Jeffrey Mfg. Co., April 15, 1925.

4. Type 56-A drilling machine; 2-type A-5 Jeffrey drills on truck, propelled by 13-hp. motor, 250-500 volts, D. C. Approvals 147 and 147A, issued to The Jeffrey Mfg. Co., February 8, 1929.

5. Model A drill; 1 3/4 hp. motor, 250 volts, D. C. Approval 154 issued to Ohio Brass Co., Aug. 1, 1928.

6. Class CD-4 and CD-5 self-propelled coal drilling machine; 5-hp. motor, 250-500 volts, D. C. Approvals 176 and 176A issued to Sullivan Machinery Co., September 9, 1929.

7. Class CD-4 and CD-5 self-propelled coal drilling machine; 5-hp. motor, 250-500 volts, D. C. Approvals 177 and 177A issued to Sullivan Machinery Co., September 13, 1929.

Mining Machines

1. Type CE-7 shortwall mining machine; 30-hp. motor, 250-500 volts, D. C. Approvals 100 and 100A, issued to Sullivan Machinery Co., Sept. 30, and Oct. 20, 1914, respectively.

2. Types 12-CC and 12-EC shortwall mining machines, 35-hp. motor, 210-500 volts, D. C. Approvals 101 and 101A, issued to Goodman Mfg. Co., May 20, 1916.

3. Type 35-B shortwall mining machine, 35-hp. motor, 250-500 volts, D. C. Approvals 103 and 103A, issued to The Jeffrey Mfg. Co., Nov. 2, 1917.

4. Type CE-7 shortwall mining machine, 30-hp. motor, 220-440 volts, A. C. Approvals 104 and 104A, issued to Sullivan Machinery Co., January 16, 1919.

5. Types 12-CJ and 12-EJ shortwall mining machines; 50-hp. motor, 210-500 volts, D. C. Approvals 105 and 105A, issued to Goodman Mfg. Co., June 21, 1920.

6. Types 112-CC and 112-EC shortwall mining machines; 50-hp. motor, 210-500 volts, D. C. Approvals 106 and 106A, issued to Goodman Mfg. Co., Feb. 9, 1922.

7. Types 12-CC and 12-EC shortwall mining machines; 35-hp. motor, 210-500 volts, D. C. Approvals 107 and 107A, issued to Goodman Mfg. Co., Feb. 9, 1922.

8. Types 112-CJ and 112-EJ shortwall mining machines; 35-hp. motor, 210-500 volts, D. C. Approvals 108 and 108A, issued to Goodman Mfg. Co., Feb. 9, 1922.

9. Type 35-BB shortwall mining machine; 50-hp. motor, 250-500 volts, D. C. Approvals 111 and 111A, issued to Jeffrey Mfg. Co., Oct. 16, 1922.

10. Type 29-C arcwall mining machine; 50-hp. motor, 250-500 volts, D. C. Approvals 112 and 112A, issued to Jeffrey Mfg. Co., March 13, 1924.

11. Types 212-EJ and 212-CJ shortwall mining machines; 50-hp. motor, 210-500 volts, D. C. Approvals 113 and 113A, issued to Goodman Mfg. Co., Nov. 4, 1924.

12. Types 112-CK3 and 112-EK3 mining machines; 35-hp. motor, 220-440 volts, A. C. Approvals 114 and 114A, issued to Goodman Mfg. Co., Feb. 7, 1925.

13. Types 112-CL3 and 112-EL3 shortwall mining machines; 50-hp. motor, 220-440 volts, A. C. Approvals 115 and 115A, issued to Goodman Mfg. Co., Feb. 7, 1925.

14. Type 124-EJ slabbing machine; 50-hp. motor, 210-500 volts, D. C. Approvals 118 and 118A, issued to Goodman Mfg. Co., March 12, 1925.

15. Type 30-A shearing-drilling machine; 50-hp. motor, 250-500 volts, D. C. Approvals 125 and 125A, issued to The Jeffrey Mfg. Co., April 26, 1926.

16. Type CLU cutting-shearing machine; 30-hp. motor, 250-500 volts, D. C. Approvals 134 and 134A, issued to the Sullivan Machinery Co., March 18, 1927.

17. Type CLE longwall mining machine; 30-hp. motor, 250-500 volts, D. C. Approvals 136 and 136A, issued to the Sullivan Machinery Co., May 28, 1927.

18. Type S-3 shearing machine; 25-hp. motor, 500 volts, D. C. Approval 148A, issued to Joy Mfg. Co., Feb. 8, 1928.

19. Type B-2 mining machine; 50-hp. motor 250-500 volts, D. C. Approvals 152 and 152A, issued to Oldroyd Machine Co., June 15, 1928.

20. Type 35-BB shortwall mining machine; 50-hp. motor, 220-440 volts A. C. Approvals 153 and 153A, issued to The Jeffrey Mfg. Co., Aug. 1, 1928.

21. Type CLE-2 longwall mining machine; 50-hp. motor, 220 volts A. C. Approval 181, issued to the Sullivan Machinery Co., Dec. 2, 1929.

Room Hoists

1. Oaks safety room hoist; 5-hp. motor, 250-500 volts, D. C. Approvals 116 and 116A, issued to South Fork Foundry and Machine Co., Feb. 13, 1925.

2. Brownie room hoist; 5-hp. motor, 250-500 volts D. C. Approvals 162 and 162A, issued to Brown-Fayro Co., Nov. 30, 1928.

3. Type HDE-4 room hoist; 10-hp. motor, 250-500 volts, D. C. Approvals 164 and 164A, issued to Sullivan Machinery Co., Jan. 4, 1929.

4. Type HE-5 room hoist; 10-hp. motor, 250-500 volts, D. C. Approvals 165 and 165A, issued to Sullivan Machinery Co., Jan. 4, 1929.

5. Brownie model HG room hoist; 5-hp. motor, 250 volts, D. C. Approval 169, issued to the Brown-Fayro Co., April 5, 1929.

Mine Pumps

1. Austin 5 by 6 mine pump; 5-hp. motor, 500 volts, D. C. Approval 121A, issued to Dravo-Doyle Co., Oct. 1, 1925.

2. Austin 5 by 6 mine pump; 5-hp. motor, 250-500 volts, D. C. Approvals 124 and 124A, issued to Dravo-Doyle Co., April 14, 1926.

3. Fairmont No. 1 mine pump; 5-hp. motor, 115, 230, and 500 volts, D. C. Approvals 140 and 140A, issued to Westinghouse Electric & Mfg. Co., Nov. 1, 1927.

4. Deming "Oil Rite" mine pump; 5-hp. motor, 115, 230, and 500 volts D. C. Approvals 141 and 141A, issued to Westinghouse Electric & Mfg. Co., Nov. 1, 1927.

5. Scranton 5 by 6 mine pump; 5-hp. motor, 115, 230, and 500 volts, D. C. Approvals 143 and 143A, issued to Westinghouse Electric & Mfg. Co., Nov. 1, 1927.

6. Weinman "Self-Oiler" mine pump; 5-hp. motor, 115, 230, and 500 volts, D. C. Approvals 144 and 144A, issued to Westinghouse Electric & Mfg. Co., Nov. 1, 1927.

7. Austin 5 by 6 mine pump; 5-hp. motor, 115, 230, and 500 volts D. C. Approvals 145 and 145A, issued to Westinghouse Electric & Mfg. Co., Nov. 18, 1927.

8. Austin 4 by 5 mine pump; 3-hp. motor, 230 volts, D. C. Approval 157, issued to General Electric Co., Aug. 23, 1928.

9. Austin 5 by 6 mine pump; 5-hp. motor, 230-500 volts, D. C. Approvals 158 and 158A, issued to General Electric Co., Aug. 27, 1928.

10. Deming "Oil-Rite" mine pump; 3-hp. motor, 230 volts, D. C. Approval 161, issued to General Electric Co., Nov. 9, 1928.

11. Deming "Oil-Rite" mine pump; 5-hp. motor, 230 volts, D. C. Approval 163, issued to General Electric Co., Dec. 7, 1928.

12. Scranton 5 by 6 mine pump; 5-hp. motor, 250-500 volts, D. C. Approvals 170 and 170A, issued to the Goodman Mfg. Co., April 9, 1929.

Concrete Mixers

1. Austin-type concrete mixer; 5-hp. motor, 115, 230, and 500 volts, D. C. Approvals 142 and 142A, issued to Westinghouse Electric & Mfg. Co., Nov. 1, 1927.

Rock-Dusting Machine

1. M.S.A. Co. rock-dusting machine; 5-hp. motor, 230 volts, D. C. Approval 130, issued to Mine Safety Appliances Co., November 5, 1926.

2. Diamond Machine Co. rock-dusting machine; 15-hp. motor, 230 volts, D. C. Approval 131, issued to Diamond Machine Co., December 28, 1926.

3. M.S.A. Co. rock-dusting machine; 15-hp. motor. 230 volts, D. C. Approval 137, issued to Mine Safety Appliances Co., July 2, 1927.

4. Type 125 rock-dusting machine; 12-hp. motor, 250-500 volts D. C. Approvals 146 and 146A, issued to Mine Safety Appliances Co., Jan. 20, 1928, and April 3, 1928, respectively.

5. Type H rock-dusting machine; 20-hp. motor, 230 volts D. C. Approval 180, issued to The American Mine Door Co., Oct. 30, 1929.

PERMISSIBLE ELECTRIC SWITCHES AND JUNCTION BOXES

Approved Under Schedule 4A

1. Enclosed 2-pole fused switch-200 amperes, 250 volts; 100 amperes, 500 volts, D. C. Approvals 400 and 400A, issued to The Ohio Brass Co., June 16, 1928, and Aug. 5, 1925, respectively.

2. Enclosed 2-pole fused switch - 250 amperes, 250 volts; 150 amperes, 500 volts, D. C. Approvals 401 and 401A, issued to the Sullivan Machinery Co., May 11, 1927.

3. Enclosed 3-pole fused switch - 250 amperes, 220 volts; 150 amperes, 440 volts, A. C. Approvals 402 and 402A, issued to the Sullivan Machinery Co., May 11, 1927.

PERMISSIBLE ELECTRIC CAP LAMPS FOR MINERS

Approved Under Schedules 6A & 6B

1. Edison Model "C" lamp. Approval 10, issued to Edison Storage Battery Co., Feb. 24, 1915.

2. Wico lamp, Approval 14, issued to Witherbee Igniter Co., June 10, 1916.

3. Wheat lamp. Approval 17, issued to Koehler Mfg. Co., Inc., Sept. 23, 1919.

4. Edison Model "E" lamp. Approval 18, issued to Edison Storage Battery Co., March 28, 1923.

5. RM-6 f. d. Ceag lamp. Approval 19, issued to Concordia Electric Co., Aug. 2, 1923.

6. Super-wheat lamp. Approval 20, issued to Koehler Mfg. Co., (Inc.), April 27, 1926.

7. RM-7 Ceag lamp. Approval 21, issued to Concordia Electric Co., June 18, 1926.

8. Edison Model G lamp. Approval 22, issued to Edison Storage Battery Co., Nov. 16, 1927.

9. RM-8 Ceag lamp. Approval 23, issued to Concordia Electric Co., April 3, 1928.

PERMISSIBLE FLAME SAFETY LAMPS

Approved under Schedules 7, 7A, and 7B

1. Koehler steel frame lamp, flat wick. Approval 201, issued to Koehler Mfg. Co., (Inc.), Aug. 21, 1915.

2. Koehler Steel frame lamp, round wick. Approval 201A, issued to Koehler Mfg. Co., (Inc.), July 29, 1918.

3. Koehler Aluminum frame lamp, flat wick. Approval 203, issued to Koehler Mfg. Co., (Inc.), Feb. 7, 1919.

4. Koehler Aluminum frame lamp, round wick. Approval 203A, issued to Koehler Mfg. Co., (Inc.), Feb. 7, 1919.

5. Wolf brass frame lamp, round wick. Approval 204, issued to Wolf Safety Lamp Co. of America, (Inc.), July 18, 1921.

6. Wolf Aluminum frame lamp, round wick. Approval 205, issued to Wolf Safety Lamp Co. of America, (Inc.), April 24, 1924.

7. Wolf Aluminum frame lamp, flat wick. Approval 206, issued to Wolf Safety Lamp Co. of America, (Inc.), April 24, 1924.

8. Wolf brass frame lamp, flat wick. Approval 208, issued to Wolf Safety Lamp Co. of America, (Inc.), March 14, 1927.

PERMISSIBLE ELECTRIC HAND AND TRIP LAMPS

Approved under Schedule 10A

1. Type RMC-RMCT Ceag hand and trip lamp. Approval 1000, issued to Concordia Electric Co., May 25, 1922.

2. Model "E" inspection lamp. Approval 1001, issued to Mine Safety Appliances Co., July 28, 1925.

3. Model "E" signal lamp. Approval 1002, issued to Mine Safety Appliances Co., December 8, 1927.

4. Model "E" hand lamp. Approval 1003, issued to Mine Safety Appliances Co., Jan. 12, 1928.

5. Super-Wheat hand lamp. Approval 1004, issued to Koehler Mfg. Co., (Inc.), July 19, 1928.

6. Super-Wheat Stevedore Lamp. Approval 1005, issued to Koehler Mfg. Co., (Inc.), August 3, 1928.

PERMISSIBLE ELECTRIC FLASH LAMPS

Approved Under Schedule 11

1. Eveready, safety type, flash lamp. Approval 601, issued to National Carbon Co., (Inc.), Oct. 22, 1924.

PERMISSIBLE METHANE INDICATORS AND DETECTORS

Approved Under Schedules 7B and 8A

Methane Indicating Detectors

1. Burrell indicator. Approval 800, issued to Mine Safety Appliances Co., March 10, 1922.

2. "U.C.C." indicator. Approval 802, issued to Oxweld Acetylene Co., Nov. 27, 1928.

Methane Detectors

1. Wolf flame-type detector. Approval 207, issued to Wolf Safety Lamp Co. of America, (Inc.), Nov. 21, 1924.

2. Martienssen electric-type detector. Approval 801, issued to the Gesellschaft fuer nautische Instrumente, Kiel, Germany, January 9, 1928.

PERMISSIBLE MINE TELEPHONES

Approved Under Schedule 9A

1. Western Electric mine-type telephone. Approval 901, issued to the Western Electric Co., (Inc.), July 16, 1927.

PERMISSIBLE SINGLE-SHOT BLASTING UNITS

Approved Under Schedule 12

1. M.S.A. battery-type blaster. (Edison Model "C" mine lamp battery with blasting attachment.) Approval 1200, issued to Mine Safety Appliances Co., May 24, 1924.

2. Davis No. 0 magneto-type blaster. Approval 1201, issued to Davis Instrument Mfg. Co., (Inc.), March 15, 1921.

3. Du Pont Pocket magneto-type blaster. Approval 1202, issued to E. I. Du Pont de Nemours & Co., Aug. 15, 1924.

4. Davis No. 00 magneto-type blaster. Approval 1203, issued to Davis Instrument Mfg. Co. (Inc.), Oct. 17, 1924.

5. Concordia battery-type blaster. (Concordia type RM-6 f.d. mine lamp battery with blasting attachment.) Approval 1204, issued to the Concordia Electric Co., March 2, 1925.

6. M.S.A. battery-type blaster. (Edison Model "E" mine lamp battery with blasting attachment). Approval 1205, issued to the Mine Safety Appliances Co., April 28, 1925.

7. Eveready dry cell-type blaster. Approval 1206, issued to National Carbon Co., (Inc.), Aug. 20, 1925.

8. Davis No. 000 magneto-type blaster. Approval 1207, issued to Davis Instrument Manufacturing Co., (Inc.), Nov. 18, 1926.

PERMISSIBLE STORAGE BATTERY LOCOMOTIVES AND POWER TRUCKS

Approved Under Schedule 15

Gathering Locomotives

1. Whitcomb, E.S.B. flame-proof locomotive. Approval 1500, issued to Geo. D. Whitcomb Co., March 14, 1921.

2. Jeffrey type B.D.H. class 40 locomotive. Approval 1501, issued to The Jeffrey Manufacturing Co., October 11, 1921.

3. Mancha flame-proof "Hercules" locomotive. Approval 1502, issued to the Mancha Storage Battery Locomotive Co., Nov. 13, 1922.

4. Ironton type W.O.G. locomotive. Approval 1503, issued to Ironton Engine Co., March 24, 1923.

5. Mancha Hercules A and AX locomotives. Approval 1505, issued to the Mancha Storage Battery Locomoti ve Co., April 5, 1924.

6. Jeffrey type B.D.M. Class 25 locomotive. Approval 1507, reissued to the Jeffrey Mfg. Co., Aug. 20, 1925.

7. Goodman Type 10-30 locomotive. Approval 1508, issued to the Goodman Mfg. Co., March 21, 1925.

8. Goodman Type 8-30 locomotive. Approval 1509, issued to the Goodman Mfg. Co., Sept. 25, 1925.

9. Mancha Standard A. AN, and AX locomotives. Approval 1511, issued to Mancha Storage Battery Locomotive Co., Nov. 10, 1925.

10. Westinghouse-Baldwin locomotive. Approval 1512, issued to Westinghouse Electric & Mfg. Co., Nov. 11, 1925.

11. General Electric type LSBE-2C6-C9 locomotive. Approval 1513, issued to General Electric Co., February 25, 1926.

12. Jeffrey Type B.D.M. 25 Form H. Locomotive. Approval 1516, issued to The Jeffrey Mfg. Co., December 28, 1926.

13. Atlas Type B locomotive. Approval 1517, issued to Atlas Car & Mfg. Co., Feb. 10, 1927.

14. Mancha 2-motor locomotive; Approval 1520, issued to the Mancha Storage Battery Locomotive Co., May 27, 1929.

Main-Line Haulage Locomotive

1. Jeffrey Type B.D.M. Class 30 main-line haulage locomotive. Approval 1510, issued to Jeffrey Mfg. Co., Oct. 12, 1925.

Power Trucks

1. Mancha power tank. Approval 1506, issued to Mancha Storage Battery Locomotive Co., May 5, 1924.

2. Mancha power tank and gathering locomotive. Approval 1505A, issued to Mancha Storage Battery Locomotive Co., June 21, 1926.

3. Jeffrey power truck and main-line haulage locomotive. Approval 1510-C, issued to The Jeffrey Manufacturing Co., Dec. 31, 1926.

4. Mancha nonpropelled power truck, Approval 1514, issued to Mancha Storage Battery Locomotive Co., December 18, 1926.

5. Jeffrey power truck. Approval 1515, issued to The Jeffrey Mfg. Co., December 28, 1926.

6. Westinghouse power truck. Approval 1512-C, issued to Westinghouse Electric and Mfg. Co., Sept. 13, 1928.

7. General Electric type LSBE-2C6-C12 power truck and gathering locomotive. Approval 1519-C, issued to the General Electric Co., April 6, 1929.

Tandem Locomotives

1. Jeffrey tandem locomotive. Approval 1518, issued to the Jeffrey Mfg. Co., Nov. 21, 1927.

PERMISSIBLE SELF-CONTAINED OXYGEN BREATHING APPARATUS AND GAS MASKS

Approved Under Schedules 13 and 14A

Oxygen Breathing Apparatus

1. Gibbs mine rescue breathing apparatus. Approval 1300, issued to Mine Safety Appliances Co., January 15, 1920.

2. Paul mine rescue breathing apparatus. Approval 1301, issued to American Atmos. Corp., January 15, 1920.

3. Fleuss-Davis Proto mine rescue breathing apparatus. Approval 1302, issued to Siebe, German and Co., (Ltd.), February 7, 1924.

4. McCaa mine rescue breathing apparatus. Approval 1303, issued to Mine Safety Appliances Co., Aug. 31, 1925.

5. Draeger mine rescue breathing apparatus. Approval 1304, issued to Draegerwerk, Lubeck, Germany, Aug. 1, 1929.

Gas Masks

1. M.S.A. ammonia gas mask. (Formerly the Burrell Ammonia mask). Approval 1401, issued to Mine Safety Appliances Co., April 10, 1920.

2. M.S.A. self-rescuer. Approval 1402, issued to Mine Safety Appliances Co., March 6, 1924.

3. All-service gas mask. Approval 1403, issued to Mine Safety Appliances Co., July 1, 1925.

4. G.M.D. ammonia gas mask. Approval 1404, issued to Mine Safety Appliances Co., March 10, 1926.

5. La France ammonia gas mask. Approval 1401, issued to American La France and Foamite Industries, (Inc.), June 23, 1927.

6. G.M.D. ammonia gas mask. Approval 1406, issued to Mine Safety Appliances Co., Feb. 14, 1928.

7. All-service gas mask. Approval 1405, issued to Mine Safety Appliances Co., Aug. 24, 1928.

8. Davis ammonia gas mask. Approval 1408, issued to Bullard-Davis, (Inc.), Dec. 19, 1928.

9. M.S.A. Combination hose mask. Approval 1901, issued to Mine Safety Appliances Co., May 29, 1929.

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I. C. 6231 March 1930

DEPARTMENT OF COMMERCE
UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING LAWS OF FRANCE

BY

E. P. YOUNGMAN

I. C. 6231.
March, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF FRANCE[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. In preparing this interpretation of the laws of France extensive use has been made of an analysis prepared by V. Demir, forwarded by Daniel J. Reagan, American commercial attaché, Paris, through the courtesy of the Department of Commerce.

INTRODUCTION

The basic mining law of France is the Mining Law of April 21, 1810, modified by the subsequent laws of May 9, 1866, July 21, 1880, July 23, 1907, September 9, 1919, December 16, 1922, and numerous other laws and regulations generally of minor importance. No attempt has been made to point out regulations that apply to mines secured before the passage of the 1919 law when they differ from those relating to concessions obtained since, as the Law of 1919 affects American interests.

The portions of Alsace-Lorraine ceded by Germany to France in 1918 continue to be governed by the "local laws," that is, by the German mining laws in effect in Alsace-Lorraine in 1918. The fundamental German mining law is the law of December 16, 1873, modified by the laws of December 8, 1909, August 23, 1912, and June 25, 1913. The German laws in effect apply mainly to prospecting, rights of the concessionaire, conveyance of concessions, and damages to surface landowners. The French Law of December 9, 1919, and the regulations pertaining thereto are nevertheless effective in Alsace-Lorraine under certain temporary restrictions.

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1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6231."

2 Rare metals and nonmetals division.

CLASSIFICATION

Mineral resources are classed under three distinct categories: Quarries, "minières," and mines proper.

1. Quarries comprise such deposits as slate, sandstone, sand, granite, lava, basalt, marble, limestone, chalk, gypsum, flint, kaolin, clay, freestone, gravel, etc., exploited in the open or by galleries. (Sec. 4, Law of 1810.)

2. Minières include alluvial iron deposits, pyritic earths convertible into iron sulphate, aluminous or siliceous deposits, and peat, as well as metallic deposits operated in the open. (Sec. 57, Law of 1810.)

3. Mines are those deposits that are known to contain, either in veins, lodes, strata, or otherwise, such mineral substances as gold, silver, platinum, mercury, lead, iron in veins or in strata, copper, tin, zinc, calamine, bismuth, cobalt, arsenic, manganese, antimony, molybdenum, and other metallic deposits, graphite, sulphur, coal, anthracite, bitumen, alum, metallic sulphates, rock salts, petroleum, and natural gas. (Sec. 2, Law of 1810.) This enumeration is not complete; other minerals may be added to this list by decree in Council of State or by a law, as in the case of rock salt (Law of June 17, 1840) and of petroleum and natural gas (Law of December 16, 1902).

In any case, it is the nature of the deposit that determines its character as a mine, a minière, or a quarry. Alluvial iron deposits exploited by underground methods are considered mines, whereas iron-ore deposits in veins operated by open pits are classed as minières. (Sec. 57, Law of 1810, modified by Law of May 9, 1866.) Oil wells are classed as mines.

OWNERSHIP

When France was a monarchy, ownership of the surface was looked upon as giving title to everything that lay beneath the land, but at the present time mines are considered the property of the State. Although the Law of September 9, 1919, does not state explicitly that mines, even those located on private property, belong to the State, it does omit all expressions in the wording of the Law of 1810 that might be construed as the negation of the rights of the State to mines and mineral resources, and further it modifies the latter law in two important respects, as follows:

1. Mining concessions granted under these provisions are no more considered a special form of real estate and can be granted for only a limited number of years, at the end of which, unless renewed, they return to the State as they stand at the time, together with the buildings, land, engines, machinery, and all the accessories necessary for their operation (which the concessionaire must own in full and maintain in a fair working state), without any indemnity whatsoever to the former concessionaire (pars. 2 and 4, sec. 2, Law of 1919). Nevertheless, a mining concession is

considered real property and as such can be mortgaged. The ground necessary for the operation of the mine may be purchased, leased, or "temporarily occupied." (Sec. 43, Law of 1922.) When grounds are leased, the lease must contain a substitution clause in favor of the State in the event of renunciation by the concessionaire, of revocation, or of normal termination of the concession.

2. Concessions are no more gratuitous, the State and the personnel of the mine participating in the profits thereof. The concessionaire, on the other hand, is subject to financial obligations imposed on him by the contract of concessions (Cahier des Charges).

The Cahier des Charges, which is in effect a contract between the State and the concessionaire, defines the rights and obligations of both. The Law of 1919 states only the main principles to be embodied in that contract and provides that a general decree of public administration shall draw up the typical contract, the text of which is given in the Decree of April 21, 1920, and the terms of which are those embodied in this abstract, for the most part. However, in accordance with a great principle of French law that laws provide only for the future and are not retroactive, the provisions apply only to concessions granted after September 9, 1919.

RIGHTS OF FOREIGNERS

Authorization for prospecting may be given both to companies and to individuals of French or foreign nationality. Any French citizen or any foreigner naturalized or not in France, acting individually or as a company, is entitled to solicit, and may be granted, a mining license. (Sec. 13, Law of 1810.) However, a special procedure must be followed when concessions are granted to individuals or companies of foreign nationality (Sec. 2, Decree of May 28, 1923), and decrees of concession must be countersigned by the President of the Council of Ministers, the Minister in Charge of Mines, and the Minister of Foreign Affairs. In other words, concessions may be granted to aliens only when the consent of three ministerial departments has been obtained; and in practice the provisions concerning the right of foreigners to obtain mining concessions are more or less ineffective. In the event that the Government sees fit to refuse to grant a concession it is not obligated to disclose its reasons. Only rarely are mining concessions granted either to foreign citizens or to foreign companies. No objection is made on the ground of nationality, however, to the inclusion of foreigners in French corporations, provided that the majority of the shares belong to French citizens and provided (according to section 2 of the Cahier des Charges, by the Decree of April 21, 1920, under the Law of 1919) that two-thirds of the directors are French. Apparently, there is then nothing to prevent the creation of a subsidiary mining corporation in which the majority of the stock belongs to French citizens, who may be bound by special agreements to act on the instructions of the foreign group. In no case, however, may the composition of the board of directors be modified in the sense of reducing the French majority.

PROSPECTING

General

Prospecting for mineral substances in France is not free. The right to prospect belongs in principle to the owner of the land, who may prospect without any formalities whatsoever. If the owner of the surface land does not desire to prospect, he may sell or otherwise grant that right to another. In that case the owner and the prospector are free to determine their mutual obligations; but if the prospector desires to explore land for which the proprietor will not give his consent or if he desires to explore public lands, he must obtain an authorization from the Government in the form of a decree. The Law of December 12, 1922, determines the procedure to be followed and also the mutual rights of the proprietor and the prospector, and the Decree of August 14, 1923, outlines the procedure for the instruction of petitions for prospecting. The decree authorizing a prospector to proceed is granted only after an inquiry, at which the owner of the land must be heard.

The indemnity that the prospector must pay to the proprietor of the land may be agreed upon between them; should no agreement be reached, the indemnity is determined by section 43 of the Law of 1810, as it is modified by that of December 16, 1922.

Without permission from the owner the prospector may not bore wells or make other openings within inclosed spaces or in gardens or within 50 meters from dwellings or walled grounds.

Prospecting rights may be granted to companies or to individuals of French or of foreign nationality.

The Government has discretionary power to grant or to refuse a prospecting permit and to assign limits to it.

Further provisions with respect to prospecting may be found in the two decrees of August 14, 1923.

Prospecting for Oil

Since oil is of vital importance to France and since geological facts show that petroleum may be found there, prospecting for oil is encouraged by the Government, although none of the numerous attempts that have been made to discover oil have yet had important results, except in the Pechelbroun oil fields in Alsace (in operation since the War) and in those of Gavian, in the Hérault Department, and in a few other fields.

Special provisions with respect to oil prospecting were made in the Law of December 16, 1922, which were incorporated in section 10, No. 2, Law of 1810. By those provisions, a special permit (Permis Exclusif de Recherches) is required for the prospecting of oil and natural gas.

The special permit, which is granted only after the holding of a special inquiry at which all those interested may express their opinions, gives the prospector exclusive right (exclusive even of the landowner himself) to bore within the bounds fixed by the decree of authorization. It is the only permit that gives the holder prior right to a concession later. (See p. 6.)

The area covered by an oil permit may not exceed 5,000 hectares (50 square kilometers). Although several permits may be issued to the same person or company the respective grants must be at least 10 kilometers apart. The duration of the permit is two years, but an extension for a further period not to exceed one year may be given by ministerial permission upon the recommendation of the General Council of Mines.

The prospector of oil is subject to a yearly tax of 3 francs a hectare. (See p. 11.)

EXPLOITATION

QUARRIES AND MINIÈRES

Both quarries and minières are considered private property, and their operation is free. The proprietor of a quarry is free to exploit it without any special permission; but as quarries are under the control of the public authorities for safety purposes, any person intending to operate one must file a declaration at the town hall of the commune[3] where the quarry is situated. (Secs. 81 and 82, Law of 1810, modified by the Law of July 27, 1908.) Minières follow the same general rules as quarries, except in special cases that can not be dealt with here. When a minière is exploited in the open the proprietor before beginning work must file a declaration with the préfet of the Department[4] in which the minière is situated; but if it must be operated by underground methods he must first obtain an authorization from the same official and must comply with such regulations as that official may consider necessary. (Sec. 57, Law of 1810, modified by the Law of 1866.) When alluvial iron deposits can no longer be operated in the open, a mining concession must be granted by the proper authorities. The conditions under which this may be done, both with respect to alluvial iron deposits and to other deposits classed under minières, are covered by sections 69, 70, and 71, Law of 1810. When minières are to be operated by others than the proprietors themselves the operator must pay the owner a royalty (sec. 72, Law of 1810) and must secure an authorization from the préfet. (See secs. 76 and 77, of same law.) Peat deposits may be exploited by the proprietor of the land or by another with his consent, but a declaration must be made and an authorization must be obtained from the sous-préfet.[5] (Secs. 83 to 86, Law of 1810.) Special regulations for the working of peat deposits are contained in the decrees of August 13, 1911, and September 12, 1913.

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3 Smallest administrative division in France.

4 A Department is one of the 90 units into which France is divided; a préfet is the highest executive official of a Department.

5 A sous-préfet is the chief executive of a sous-préfeture, which is one of the three or four units into which a Department is divided.

MINES PROPER

(Underground Mines and Oil Wells)

Concessions

Priority of Right to a Concession

Neither the proprietor of the surface ground nor the discoverer of the mine has any right of preference in obtaining a mining concession, except an oil prospector. The Government has discretionary power to grant or to refuse a mining right or to give preference to one petitioner over others.

The holder of a special permit to prospect for oil and natural gas, has a definite right to the concession, provided that he has (within the period assigned for prospecting) proved the existence of liquid or gaseous hydrocarbons, and provided that he accepts the conditions of the Cahier des Charges. Section 28 of the Cahier des Charges must contain a clause imposing upon the concessionaire the obligation of making certain minimum yearly developments. (See also p. 5.)

Area of a Concession

The decree of concession determines the area and boundaries of a mine in each instance. Several concessions may be granted to one concessionaire, but each must be actually exploited. (Secs. 22 to 31, Law of 1810, modified by Law of 1880.) There are no extralateral rights.

Application Formalities

Petition and accompanying documents.--The petition, which must state whether the petitioner intends to exploit the concession personally or whether he contemplates the creation of a special company, must set forth the following details:

1. The name, profession, nationality, and domicile of the applicant, and if the petition is made in the name of a company or of a corporation, the names and nationality of the members of the board of directors, managers, or other executive officials.

2. The nature of the deposits that are to be exploited.

3. The area and the boundaries, as well as the communes (boroughs) in which the concession lies.

4. The amount of royalty offered to the proprietors of the surface.

5. The amount of the indemnity demanded by the discoverer, who is also the petitioner, in case he is not granted the concession, or the amount offered to the discoverer, who is not the petitioner, in case the petitioner receives the concession.

6. The concessions for similar substances, which the petitioner may already hold either as a concessionaire or as a lessee, and which he intends to unite with the new concession or to merge in the same company.

The following documents must accompany all petitions:

1. A large-scale map (1/80,000) of the regions in which the concession is situated.

2. A large-scale map (1/10,000 topographical map, in triplicate, of the concession asked for.

3. A statemtnt showing the extent and methods of prospecting and the results obtained.

4. Documents to prove that the petitioner is technically and financially able to develop and exploit the concession, as well as to meet the obligations that may be imposed on him by the decree of concession with respect to royalties and indemnities.

5. The proof of the nationality of the petitioner.

6. A copy of the articles of association and of the by-laws of the company or corporation, as well as the power of attorney conferred by the company or corporation on the petitioner.

Authorities.--The petition for a mining concession is sent to the prefet of the Department were the mine is located, who transmits it to the chief engineer of mines of the Department, who returns it later, with his report, to the prefet, whereupon the prefet orders an inquiry, which lasts two months. (Should the petition concern minerals that can not be considered or that are in mining concessions already granted, the minister of public works is called upon to decide whether or not an inquiry shall be held.) At the end of the two months allowed for inquiry the whole file of the case (which will include any protests, or oppositions, and any bids for preference that may have been made within that period) is again sent to the chief engineer of mines for final inspection and verification of the plans. The chief engineer sends the file, with his own report to the Minister of Public Works, who after consulting the General Council of Mines must, within three months of the period assigned to him for decision, submit to the Council of State a draft decree, granting or refusing the petition. An additional contract (Cahier des Charges) is annexed to the draft decree. (See p. 3.)

The Minister of Public Works must make his decision within a year of the close of the inquiry when there are no protests or bids for preference, otherwise within 18 months.

Qualifications of applicants for concessions.--Applicants for concessions must prove financial ability to begin and continue the working of the mines, as well as to pay the royalties, taxes, etc., and must offer surety for all indemnities in case of accidents.

Protests (oppositions) and bids for preference.--Protests or bids for preference may be made to the prefet during the 2-month period of inquiry, or they may be made to the Minister of Public Works during a period of six months.

Publicity.--Notices concerning a petition are published, during the 2-month period set for the inquiry mentioned above, in the Journal Officiel, in at least one local paper, in the administrative center of the Department (sous-prefeture), and in the communes in the territory in which the mine is located. Notice of the end of the inquiry is published in the Journal Officiel and in at least one local paper. The final decree of concession, with the Cahier des Charges, is published in the Journal Officiel and in the Bulletin des Lois; the decree is published likewise in each of the communes. The same procedure is followed with reference to protests and bids for preference.

Duration of Grant

The duration of mining concessions is now limited, as follows: 99 years for coal, lignite, petroleum and other natural fuels, and iron; 50 years for graphite, platinum, tungsten, molybdenum, bismuth, cobalt, cadmium, vanadium, and radium; and 75 years for all other deposits.

Before the beginning of the twenty-fifth year preceding the termination of the concession, the Minister of Public Works decides whether a concession shall be renewed; before the beginning of the twenty-sixth year, the concessionaire must request the minister, by registered letter, to inform him with respect to the State's intentions as to recaption. Should the concessionaire neglect to send his request at the proper time the minister notifies him of his decision within a year of the date upon which he does make his request, which must, however, be made before the beginning of the sixth year preceding the termination of the concession. If the concessionaire is not notified of an adverse decision within the required period he may consider his concession renewed for a further period of 25 years. In other words, no notification is required in case of renewal; but in case of refusal the concessionaire must be notified. (Sec. 5, Cahier des Charges.)

Restrictions as to Minerals to be Exploited

Concessions are granted for specific minerals, and the concessionaire is not entitled to exploit other deposits than those mentioned in the decree of concession. If other minerals are found on a concession and the operator desires to extend his operations over them he must seek a new grant, which the Government may refuse, according it to another.

Work Required from a Concessionaire

Beginning from the fifth year preceding the termination of a concession the concessionaire must execute (for the account and at the expense of the State) all measures relating to the maintenance and development of the concession prescribed by the chief engineer of the Bureau of Mines. Before the first day of May of each of the last five years the chief mining engineer notifies the concessionaire of the schedule of work requested by the State for the following year; the schedule must not allow the production of the mine to fall below the average yearly production of the previous five years, less 10 per cent. Contracts with respect to the execution of the work required by the State must be submitted to the chief engineer of mines of the Department in which the mine is located and are valid only after he has approved them.

The Cahier des Charges provides special measures to guard against carelessness on the part of the concessionaire during the last 25 years of the concession, should the State decide to use its right of recaption. Therefore, part of the amortization of the expenses in connection with the installation of the mine and with new permanent works developed during the last period of 25 years is borne by the State. (Secs. 6, 8, and 20, Cahier des Charges.) A special account of these expenses is drawn up every year, and at the termination of the concession the State reimburses the former concessionaire within a year for his participation in those expenses. Evaluations for the cost of labor must make necessary provisions for the overhead charges of the concessionaire. The account must be submitted to the administration before the first of April of the year following that in which the expenses were made. When the account has been verified by the Bureau of Mines of the Department, within a month of its presentation nine-tenths of the amount is paid to the concessionaire, the other tenth being paid within the month following the final approval of accounts.

The amount of annual work requested by the State may not exceed 40 per cent of the average yearly liquid working capital of the mining company, calculated on the previous five years. (Secs. 7 to 12, Cahier des Charges.)

Renunciations

The conditions under which a concessionaire may renounce his concession wholly or in part are set forth in section 13 of the Cahier des Charges. To be valid, renunciations must be accepted by the administration, and they may be admitted only when all mortgages have been cleared. The respective rights of the State and the concessionaire are settled after the renunciation has been accepted.

Revocations

A concession may be revoked under the following conditions: When a concessionaire fails to pay mine taxes or his share in the expenses made in the common interest of several adjoining concessions or the cost of works authorized by the State (see preceding paragraphs); when he refuses or neglects to execute the work requested of him during the last five years of the concession (see preceding paragraphs); when he sells, leases, or merges his concession without the previous consent of the State; and when he fails to comply with safety and police regulations or with the obligations set forth in section 28 of the Cahier des Charges.

A warning is first addressed to the delinquent, giving him a minimum period of six months in which to fulfill his obligations. If he fails to comply with the regulations or fails to give a satisfactory explanation a ministerial decision may declare the concession forfeited. The concessionaire may appeal to the Council of State within the 2-month period following the notification of revocation. If and when the revocation is finally made definite it is published in the Journal Officiel, and the concession is declared to be the property of the State.

Two months after the publication of notices the concession is put up at public auction; the net proceeds thereof, after payment to the State of dues, costs, and penalties, are paid to the former concessionaire, who loses all title to the concession.

Rights of Surface Owners

The act of concession determines, once and for all, the rights of the proprietor of the ground and those of the discoverer of the mine, and no legal action can be taken thereafter in this respect. (With regard to concessions granted before the passage of the Law of 1919, section 19 of the Law of 1810 provided that, when a mine was conceded to the proprietor, the right of property to the mine was distinct from that of the surface ground. Naturally, this provision is still valid as far as the older concessions are concerned.)

Rights and Obligations of Concessionaires

A concessionaire is authorized to occupy the ground that is necessary for the exploitation of the mine, the proprietor of the surface being entitled, of course, to a certain indemnity. The act of concession in each case determines the rights of the owner of the surface.

By a declaration of public utility, authorized by a decree of the Council of State, the concessionaire may expropriate all the ground required for the construction of railways, pipe lines, canals, roads, airways, drainage tunnels, and like services, under the provisions of the expropriation law of May 3, 1841.

The payment of the proportional tax (see p. 11) may be suspended for a certain period in favor of individual concessions, either by a special provision of the act of concession or by a decree in Council of State, if unforeseen difficulties in operation of the mine are encountered or a serious accident occurs during the exploitation of the mine. For this purpose an additional tax of 10 per cent of the proportional tax is collected to provide for a special common fund to meet the loss resulting to the State from such exonerations. (Secs. 32 to 42, Law of 1810.)

When the working of a mine floods a neighboring concession or deprives it of the water necessary for its operation an indemnity (the amount of which is estimated by experts) is due from the concessionaire of the first mine in reparation of the damage done. (Secs. 43 to 46, Law of 1810, modified by Law of 1923.)

Administrative Control

Mines are under the control of the engineers of the Bureau of Mines responsible to the préfet of each Department and to the Minister of Public Works. These engineers supervise the exploiting of the mine and act as advisers to the concessionaire. Upon their report concerning the violation of safety and police regulations, concessions (as well as authorizations for prospecting) may be suspended or revoked. (Secs. 47 to 50, Law of 1810, modified by the Law of July 23, 1907.) (See pp. 9-10.)

TAXES

The proprietors of mines granted before the law of September 9, 1919, had to pay, and still have to pay, annual taxes based upon the area of their concessions (10 francs a square kilometer), as well as a proportional tax (redevance proportionelle), calculated upon the production of the mine, which in no instance may exceed 5 per cent of the net product. Proprietors of concessions granted since the passage of the law of 1919 also must pay these taxes, which are applicable to all mines without exception, the exact liabilities being fixed by the decree of concession. The proportional tax may be suspended under special conditions, which have been stated on page 10. Prospectors for oil must pay a yearly tax of 3 francs. (See p. 5.)

SPECIAL FEATURE OF THE FRENCH LAW, INCLUDING ROYALTY

One of the characteristic features of the French law (Law of September 9, 1919, and Decree of April 21, 1920) is the provision for the participation of the State and of the personnel of the mine in the profits of the concession, the conditions of the participation being embodied in sections 15 and 16 of the Cahier des Charges. This compulsory participation of the personnel in the profits of mining enterprises, new and almost socialistic in principle, became embodied in the law of France as a result of the efforts of the General Confederation of Labor (Confederation Générale du Travail), most of whose members are affiliated with the French Unified Socialistic Party, and testifies to the strength and unity of the Federation of Miners.

The proportionate share of the State and the personnel of the mine is calculated on the profits of the mine after the deduction of the interest due to the concessionaire on the working capital; this rate of interest is that of the actual average interest accruing, during each period, to the perpetual State bonds having the highest nominal interest, plus 2 per cent. (Secs. 15 and 16, Cahier des Charges.)

The interest thus reserved to the concessionaire is cumulative. Once its amount is deducted from the net profits of the mine, the State and the personnel participate in the remaining profits, according to a progressive schedule, which varies from 10 to 75 per cent, calculated on the successive fractions of the excess profits, compared to the working capital of the company.

Section 27 of the Decree of April 21, 1920, putting into effect section 1 of the Law of September 9, 1919, provides that 25 per cent of the portion of the excess profits accruing to the State and the personnel belongs to the latter, the remaining 75 per cent being the property of the State. In principle, the portion of the profits accruing to the personnel is turned over to the Miners' Pension Fund (officially recognized by the Government), unless more than 20 per cent of the total personnel of the mine demand, by means of a petition addressed to the Minister of Public Works, that these profits be divided among them.

The portion of the profits accruing to the State (75 per cent) has no special destination and is carried on the budget as State revenue for the Department of Public Works.

Section 17 provides that the capital that will be taken as a basis for the calculation of the preferred cumulative interest, as well as the portion of the profits accruing to the State and the personnel, comprises the actual liquid working capital, plus the amount of capital invested in the organization of the mine--in other words, the money spent in prospecting and exploring the deposit, as well as that spent for the creation of the mining company, the construction of works and installations, the purchase of the machinery and equipment necessary for the operation of the mine and of the secondary works pertaining thereto, minus the amortization, which is deducted from the gross working profits, in accordance with the rules set forth in section 18. The losses of previous years are also added to the capital expenses and are amortized in the same manner.

According to section 18 the net profits are defined as being the difference between the gross working returns and the results of commercial and industrial operations in general on the one hand and the operating expenses plus the amortization on the other. The gross working returns represent the amount of the total annual sales of the products of the mine and of the secondary works attached thereto, special provision being made for calculation of these products if they are not sold at their regular value. The operating expenses comprise only the administrative, commercial, fiscal, and other ordinary expenses in general, all capital expenses for the purchase of machinery, etc., being added to the capital for subsequent amortization. The amount of the annual amortization is also added to the operating expenses. In principle, this amount must be equal to one-tenth of those fixed assets that are considered as capital expenses, or, in other words, to one-tenth of the fixed asset that must return to the State at the end of the concession. However, the Minister of Public Works may authorize a more rapid amortization. The amortization of the other fixed assets, which do not return to the State, is provided for in the portion of the profits belonging to the concessionaire.

In the case of concessions for petroleum and natural gas, the amount of the annual amortization is equal to 20 per cent of the capital expenses.

MISCELLANEOUS LABOR LAWS

The concessionaire must work out a schedule of the minimum wages paid to miners and employees. In the absence of collective labor contracts or of a direct agreement between the concessionaire and the workers a committee of 6 members, comprising 3 delegates elected by the personnel and 3 delegates chosen by the concessionaire, must be appointed to draw up a minimum salary schedule under the supervison of the chief engineer of the Bureau of Mines. Special measures are provided in the event of a conflict between the members of the committee. In case of disagreement 3 new members, chosen by the committee or by the permanent delegates of the Central Commission of Mines, at the Ministry of Public Works, are added to the committee, which then prepares the minimum wage schedule. This schedule is posted by the concessionaire, who must further communicate to the members of the wage committee all documents proving that its decisions are actually being applied. (Sec. 29, Law of April 21, 1920.)

ARBITRATION

Any dispute between the concessionaire and the State must first be submitted to a committee of arbitration, consisting of three members, the first chosen by the State from the engineers of the Bureau of Mines, the second by the concessionaire, and the third by the first two. This committee must make a decision within two months of its appointment; and should its decision not be accepted by either the State or the concessionaire, the conflict is then submitted to the administrative courts for final decision. (Sec. 31, Law of 1920.)

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MINING METHODS AND COSTS AT THE HECLA AND STAR MINES, BURKE, IDAHO.

BY

CHARLES H. FOREMAN

I. C. 6232.
February, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS AT THE HECLA AND STAR MINES, BURKE, IDAHO[1]

By Charles H. Foreman[2]

INTRODUCTION

This paper describing the mining practices of the Hecla and Star mines, both located at Burke, Idaho, is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts of the United States. About 26,000 tons per month of lead ore is produced at the Hecla mine. Costs at the mine are given for the year 1928. Stoping operations in the Star mine have covered such a short period, during which time the abnormal expenses of starting were incurred, that they are not given in this paper.

ACKNOWLEDGMENTS

The author acknowledges the assistance of James F. McCarthy, president and general manager, L. E. Hanley, general superintendent, Earl Hazelwood, mine foreman, A. C. Stevenson, electrical engineer, A. E. DeCelles, mining engineer, and of various other members of the staff of the Hecla and Star mines.

HISTORY

The Hecla lode claim was located on May 5, 1885. This claim, with the adjoining Katie May, located in 1880, formed the nucleus of the holdings of the present Hecla Mining Co.

The first development was done in the present No. 3 tunnel, the ledge being encountered within a very few feet of the portal. What prompted the starting of this work is not known by the author. At a point about 350 feet from the portal a narrow seam of galena was encountered. The present No. 1 and No. 2 tunnels were driven after the start of the lower or present No. 3 tunnel.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6232."

2 - One of the consulting engineers, U. S. Bureau of Mines; Chief engineer Hecla Mining Co.

The first dividends were paid in 1900, and since 1903 dividends have been paid practically without interruption.

The Star mine was acquired by the Sullivan Mining Co. in 1921, after the former owners had had considerable litigation over the apex of the lode. The Sullivan Mining Co. is owned equally by the Hecla Mining Co. and the Bunker Hill and Sullivan Mining Co.

GEOLOGY

The general geology of the Coeur d'Alene mining district has been covered by Ransome and Calkins[3] and by Umpleby and Jones.[4]

The main "Hecla" ore body occurs along a practically vertical shear zone in the Burke quartzite and has a course of approximately S. 45° E. Its maximum width is about 40 feet. The strike of the quartzite beds is approximately north and south and the dip is 74° to the northeast.

The Intermediate ore body occurs along a similar shear zone, parallel to the Hecla ore body. This ore body has a slight dip to the northeast and joins the original Hecla vein at about the 1,400 level, from which point downward they are mined as one continuous ore body. Both the Hecla and Intermediate veins follow a distinct gouge, along which some strike faulting has occurred. They are also generally associated with a lamprophyre dike, usually about 2 feet wide, but which in places attains a maximum width of 12 feet. The ore occurs on either or both sides of this dike. It is generally conceded that this dike had no effect upon the deposition of the ore.

The Magazine, Ore-Or-No-Go, and B Drift ore bodies occur in fissures in the wall south of the Hecla vein and have a strike of about S. 65° E. and a dip of 70 to 80° to the northeast. These ore-bearing fissures appear to intersect the Hecla vein, but have not been found on the north side of the latter. They have practically no gouge or shearing. The ore bodies are narrower than the Hecla ore body.

The Magazine is decidedly a lead ore body, while considerable amounts of zinc sulphide are found in both the Ore-Or-No-Go and B Drift ore bodies.

The East ore body occurs in a fissure east of the Hecla and has a strike of about S. 74° E. and a dip of 74° to the southwest. The shearing and gouge in the fissure is very distinct. A fault with a dip of 55° to northwest divides this ore body; the horizontal throw is from 160 feet to 75 feet and diminishes with depth. Widths of commercial ore vary from 3 to 14 feet.

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3 Ransome, F. L., and Calkins, F. C., The Geology and Ore Deposits of the Coeur d'Alene District, Idaho. U. S. Geol. Survey Prof. Paper 62, 1908, 203 pp.

4 Umpleby, J. B., and Jones, E. L., Geology and Ore Deposits of Shoshone County, Idaho. U. S. Geol. Survey Bull. 732, 1923, 156 pp.

The Wide West ore body is situated in the old Tiger-Poorman group of claims, acquired by the Hecla in 1921. The fissure in which this ore body occurs has a strike of N. 50° E. in the upper levels, gradually swinging to about S. 55° E. on the lower level. The ore, which is silver-lead-zinc, is rather narrow and occurs in a tight fissure with practically no gouge. The walls are very substantial.

The Ore-Or-No-Go is the only vein known to outcrop on the surface.

The Star mine is located upon the western extension of the Morning lode, which has been described by Ransome and Calkins.[5] It occurs along a shear zone in the Revett and upper quartzites which are cut by the fissure in both strike and dip. The ore bodies are without definite walls but pass gradually, by decrease of mineralization, into the country rock. Considerable faulting has occurred along the strike of the fissure.

The principal ore mineral in both mines is galena; a small amount of zinc is associated with the galena in the Hecla and a larger amount in the Star.

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCKS

The Hecla and Intermediate ore bodies, as stated previously, occur along a distinct shear zone with a substantial gouge and with the lamprophyre dike occurring in an irregular manner throughout the lode. There is no definite line of demarcation between lode and country rock. Stringers of ore are often found in the walls for irregular distances. Horizontal, but irregular, quartz stringers are also found running into the walls, principally in the hanging or northeast wall. Portions of the wall have a tendency to break to the quartz stringers and the gouge accompanying the fissure, which causes extremely heavy side and top pressure.

It is often necessary to mine a face having a horse of waste between two stringers of ore. This horse of waste may be 6 or 7 feet wide, but the whole face is usually mined; an effort is made only to shatter the waste to permit maximum underground sorting. This method is varied by mining one streak for two or three floors, filling tightly, and then mining the streak on the other side of the waste. The use of the latter method is determined by the width and condition of the included waste.

The mining of the Hecla and Intermediate ore bodies requires a system which allows for the extraction of ore from varying widths and at the same time provides for holding the stopes open against the extremely heavy top and side pressures.

The East ore body has walls of a more substantial nature than the Hecla, and the side and top weight is not excessive.

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5 See footnote 3.

The Magazine, Ore-Or-No-Go, and B Drift fissure veins have not been mined below the lowest (No. 3) adit. Above this level the ore bodies in the first two lodes have been mined to a point near the surface by shrinkage, which was not satisfactory due to excessive dilution. Gouge and shearing are practically absent in these fissures, and close timbering and fill should not be necessary.

The walls of the Wide West vein are very substantial and require no support.

The Star mine has conditions similar to the Hecla vein, but the ground is very badly sheared and is very "short." Widths of ore are indefinite, and walls are so soft that, in some cases, the use of sidelacing is necessary.

METHOD OF PROSPECTING AND EXPLORATION

Prospecting is done by crosscutting and drifting. As all fissure veins encountered have a strike of from 45 to 70° southeast, long crosscuts are given a general direction of S. 33° W., or the reverse.

Because of the possibility that ore-bearing fissures may occur in the south wall, this wall is crosscut at fairly frequent intervals. Short crosscuts, at right angles to the fissures, are driven from the stopes to encounter any possible branching stringers from the stopes.

When drifting and stoping, holes are drilled into the walls at frequent intervals to prospect for parallel stringers. Prospecting by long-hole drills has not been satisfactory, except for negative results obtained. Diamond drilling has not been attempted.

Any fissure or shear zone encountered by crosscuts and which has the proper strike and dip must be carefully watched, as the ore-bearing fissures have been known to "close up" in very short distances from commercial ore.

Faulting is not common and has offered no serious difficulties in prospecting.

METHODS OF SAMPLING AND ESTIMATION OF TONNAGES AND VALUES

In only recent years has a systematic method of sampling new developments been practiced. At this time all development work is watched, and when valuable mineral is encountered samples are taken at intervals, depending upon the grade and continuity of the ore. The position and width over which the sample is cut are recorded. Sufficient samples are taken in a face to show the value of the different classes or grades of material encountered in the face. After the assay returns are received these samples may be combined to give the value of parts or of all of the face, as judgment dictates.

The grade of all faces is calculated to a diluted mining width, as it is assumed that a certain portion of the wall rock will be mined with the ore and that a minimum width of stope will be carried. The amount of dilution and the minimum width of stope will be carried. The amount of dilution and the minimum width are dependent upon the condition of the vein and country rock. The tonnage and assay of the various blocks are calculated by the prismoidal formula. The factor to be used for cubic feet per ton is dependent upon the assay of the block in question.

METHODS OF DEVELOPMENT AND MINING

General Development

The Hecla mine is operated and all ore hoisted through the Hecla shaft. This shaft is located in the Canyon at Burke at the foot of a steep mountain. It is about 1,400 feet from the west end of the main ore body. Three adits enter the ore body from the surface, the lowest (the No. 3 tunnetl) being about 30 feet below the level of the collar of the shaft.

Stations at the shaft have been cut and drifts run to the Hecla main ore body on the 300, 600, 900, 1,200, 1,600, and 2,000 levels. The 1,400 level was opened the length of the ore body but was not connected with the shaft.

The main hoist formerly used (prior to November, 1929) had power for only 2,000 feet of depth. A winze was sunk to the 2,400 level near the ore body to develop the levels below the 2,000. A crosscut was driven to the lode on the 2,400 level where drifting revealed commercial ore. The winze was then sunk to the 2,800 level where crosscutting and drifting again showed ore. The development by the winze revealed enough commercial ore to justify the sinking of the main shaft and the installation of a hoist to handle the future needs of the mine.

Ore between the 2,800 and 2,000 levels after being mined will be passed through a rock raise to the 2,800 level and hauled by storage-battery locomotive to a skip pocket at the Hecla shaft. Timbers, men, and supplies for the levels not connected with the main shaft will be handled through the winze; the plan is to place these levels at 200-foot intervals.

Up to the time the 1,200 level was opened, all levels in the Hecla shaft had been placed at 300-foot intervals. At this time it was the policy to keep the stopes at practically the same height from the sill and to mine the entire length of a level at the same time in order to supply the concentrator with an average feed. Increasing widths of the ore body made this procedure rather slow on the lower levels, with the result that the chutes through the stopes to the level below were kept open with difficulty during the time of mining the upper part of the stope.

The 1,600 level was next opened, a raise driven from the 1,600 to the 1,200 level, and an intermediate drift from the raise driven upon the 1,400 level. This saved the expense of a station and of drifting from the shaft to the nearest face of the 1,400 level, and in addition gave only 200 feet of backs above the new levels. However, the double transfer of material on the 1,400 level caused considerable inconvenience, as did the repairs to ore and waste chutes between the 1,200 and 1,600 levels.

The 2,000 level was opened with the idea of taking one section of the stope, say 200 feet in length, and mining this intensely to the level above, leaving pillars of unmined ground between working stopes, these pillars to be mined after adjoining ground had been mined and filled. This method has been successful.

In starting a level a sump is cut on the station, the size depending upon the amount of water to be expected. Water is siphoned to the nearest pumping level (the 1,200 or the 2,000) below. Drifts are driven to the western limit of the ore body as the level is opened up; a raise is run to the level above after ore has been encountered. This raise serves as a second exit from the new level and provides for ventilation. It also serves as a passage for waste filling to the level when stoping has started. These raises are run from 200 to 300 feet apart, depending upon the width of the stope and the tonnage to be expected.

The East ore body was first encountered through the No. 3 tunnel, and later developed by drifts from the Hecla 600 and 900 levels. Raises were run from the 900 level to the No. 3 tunnel and drifts placed on the 750, 400, and 200 levels. One of these raises with the timber removed is used as an ore pass through the ore body to the 900 level, where the ore is hauled by motor trains to the 900 skip pocket at the Hecla shaft.

A 1,500-foot raise was also driven to the surface from the No. 3 tunnel level for ventilation and to provide another exit from the East ore body. This raise was very useful as an exit at the time of the fire of July, 1923, when the Hecla surface plant was destroyed.

All waste from levels above the 1,600 is placed in a rock raise connecting with No. 3 tunnel to be drawn for stope filling on the desired level below. This raise was driven to prove the continuity of the Ore-Or-No-Go ledge; later the timbers were removed. Waste from the 1,600 level is placed directly into the chutes leading to the 2,000-level stopes. Any waste from the 2,000 level, or below, is placed in the 2,000-level waste skip pocket, hoisted in regular skips to the waste bin on the surface, and taken by motor either into No. 3 tunnel to the top of the Ore-Or-No-Go waste raise or to the surface tramway waste bin. This latter tramway delivers excess waste from the mine and sorting plant to a point on the surface near the Hecla plant.

The Star mine is operated through a crosscut driven on the Hecla 2,000 level from the Hecla shaft. This crosscut was driven to intersect the Star ledge, upon which a drift was driven easterly to the Star-Morning end line. This level is called the Star 4,000 level.

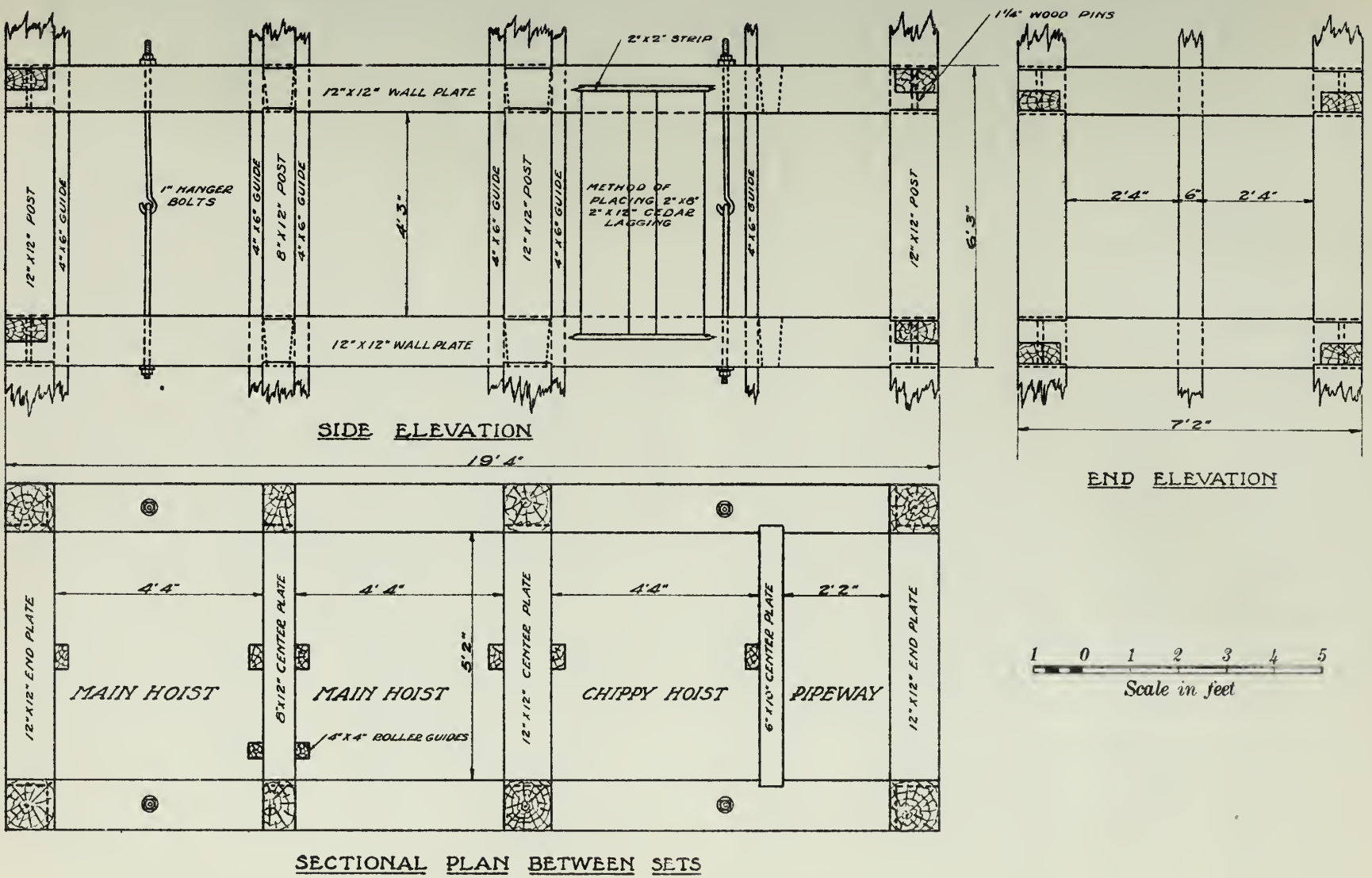


FIGURE 1. – SET OF TIMBERS, MAIN SHAFT

Considerable development was done on the upper levels of the Star mine by previous operators, but to mine the ore from the Hecla it has been necessary to work upward from the 4,000 level.

The Star 4,000 level is being connected with the upper levels by a vertical four-compartment raise, through which men and supplies are hoisted. This raise, of which 1,800 feet has been completed, will be about 2,900 feet in height when finished to the upper Star tunnel.

At present (November, 1929) the Star raise contains two hoisting compartments, one manway, and one chute. Three compartments measure 4 feet, 4 inches by 5 feet 2 inches each, the fourth being 4 feet 10-1/2 inches by 5 feet 2 inches. Timbers 12 by 12 inches and 8 by 12 inches are used, the detail being practically the same as shown in the drawing (fig. 1) of the Hecla shaft. The main hoist is located on the 2,300 level; another hoist on the 4,000 level operates a cage in the fourth compartment to take the hoistman to the 2,300-level station and for some auxiliary work. The return cable comes down one corner of the compartment.

Levels are placed at intervals of approximately 200 feet. Broken ore is drawn from the stope chutes and placed in a "rock raise" on each working level, through which it passes to the 4,000 level. It is then drawn into cars for haulage by trolley motor to the Hecla 2,000 station, where it is placed in the Star skip pocket and hoisted to the surface.

Shafts and Winze

The main shaft of the Hecla mine originally contained three compartments, two for hoisting and the third for a manway and pipes. The hoisting compartments, exclusive of guides, are 4 feet 4 inches by 5 feet 2 inches. In 1918 the third compartment containing the manway and pipes was divided so as to make a third compartment for auxiliary hoisting and yet leave sufficient room for the manway and pipes (fig. 1). The third or auxiliary compartment was used as the sinking compartment in deepening the shaft from the 2,000 to the 2,800 level; the sinking hoist was on the 2,000 level and the sinking sheave between the 1,600 and 2,000 levels. This caused the 1,600 level to be the lowest level served by the auxiliary hoist while sinking was in progress. During this period a raise in the Hecla ore body between the 1,600 and 2,000 levels was equipped with a cageway. This provided auxiliary man transportation to the 2,000 level.

Drilling in the shaft is done with sinking drills, using 1-inch, hollow-hexagon, collared drill steel. The sizes and types of bits and lengths of changes are as follows:

Size of bit, inches	Length of steel, feet	Type of bit
2-1/4	2	X-bit
2	3	Carr bit
1-7/8	4	Do
1-3/4	5	Do
1-5/8	6	Do
1-1/2	7	Do
1-3/8	8	Do
1-1/4	9	Do

The crew for sinking operations consists of seven men--one gang boss, five miners, and one hoist engineer. The cycle of operations is as follows:

Blast; clean wall plates; shovel, using two sinking buckets, one being in the bottom while the other is hoisted; when sufficient broken material has been removed, place and block timbers; shovel and clean bottom; drill; load; and blast.

Blasting is done with electric-delay primers, with the blasting box controlling the current placed on the station above. This blasting box is constructed so that it is impossible to lock the door to the box while the switch is closed. Keys to the box are carried by the boss.

The type of round drilled, averaging 44 holes, with the number of delay used in blasting each hole, is shown in Figure 2.

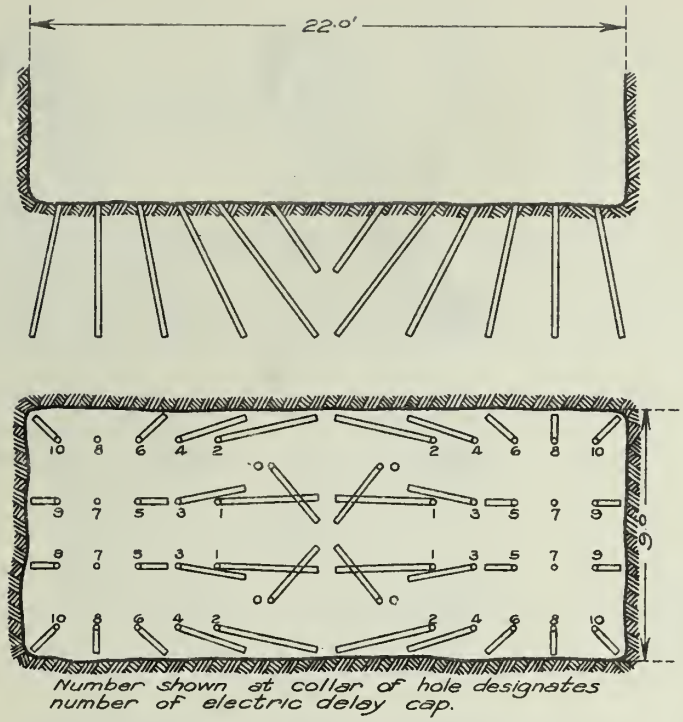
The winze driven from the 2,000 to the 2,800 level consists of three compartments, each 4 feet 4 inches by 5 feet 2 inches. Wall plates 10 by 10 inches with 8 by 10 inch dividers were used in timbering. Thirty-six holes were drilled for each round, with the same general type of round as used in the Hecla shaft.

Drifts and Crosscuts

Main haulage levels in waste are driven 8 by 8 feet in the clear to provide clearance for cars and to give sufficient cross section for ventilation. The depth of the round varies from 3 feet 6 inches to 5 feet 6 inches, depending upon the ground encountered.

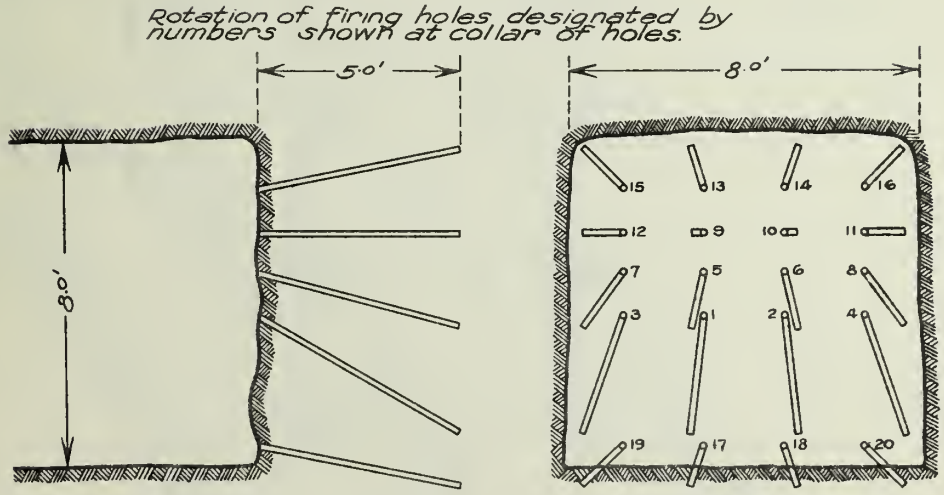
Rounds are drilled with two Leyner machines placed upon a horizontal bar. The two drills in a face are operated by three men. Hollow-round 1-1/4-inch drill steel is used, made into a cross bit of the following sizes:

Size of bit, inches	Length of steel, feet
2-1/4	3
2	5
1-3/4	7
1-1/2	9



METHOD OF PLACING HOLES IN SHAFT SINKING AT THE HECLA

FIGURE 2



STANDARD DRIFT ROUND FOR HARD GROUND AS PLACED AT THE HECLA

FIGURE 3

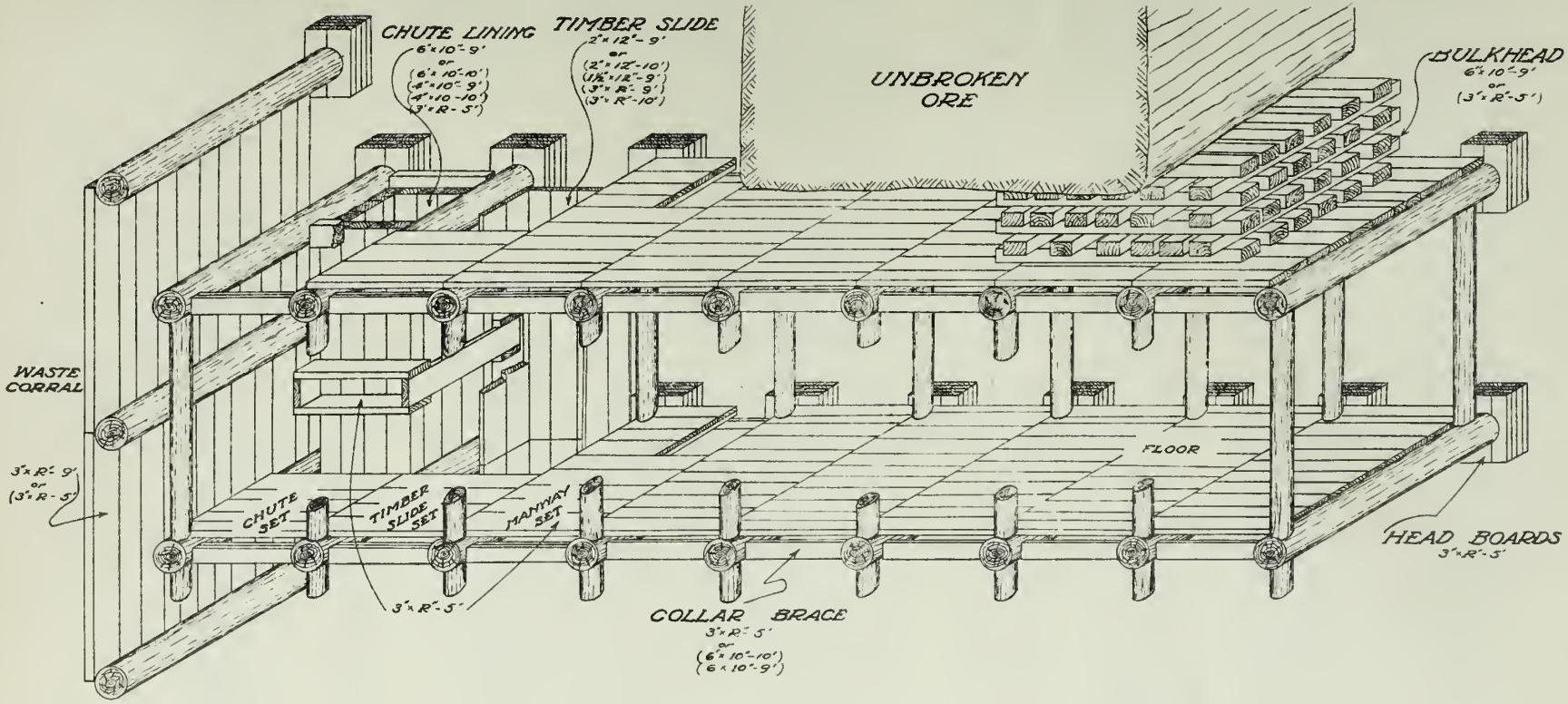


FIGURE 4.– SECTION OF STOPE SHOWING TIMBERING

0 1 2 3 4 5 10

Scale in feet

Holes are drilled in rows, five to each row, four rows being necessary in an 8 by 8 foot drift or crosscut. The type of round for this size of drift, in hard ground, is given in the accompanying sketch (fig. 3). In softer ground the second cuts may be omitted.

When ore is encountered, the height of a drift is increased to 13 feet to give stoping room above timbers. The full width of the ore is removed up to a maximum of 18 feet; a minimum width of 10 feet is taken under stopes. This additional width of face calls for extra holes, depending upon the width and hardness.

Crosscuts are generally driven 5 by 7 feet in the clear, as this is the minimum cross section for proper placing of cut holes and for the operation of a shoveling machine. In hard ground three cut holes, three second cuts, three breast holes, three back holes, and three lifters are necessary. In softer ground the second cuts are omitted.

Shoveling is done both by hand and with a shoveling machine; the machine is used where the amount of broken material is large or where speed is necessary.

Raises

Various types of raises have been driven in the Hecla mine. The usual type driven in the ore body when opening a new level contains three compartments consisting of a chute, a timber slide, and a manway. The raise is run about 18 feet long on the strike of the vein; the full width of the ore is removed to a maximum of 18 feet. The raise is timbered with stull sets. Round caps, 5 feet center-to-center measured horizontally and 9 feet measured vertically, are placed at right angles to the strike of the lode (fig. 4). Posts are used under each end of the caps. In extremely heavy ground the vertical distance between caps may be changed to 5 feet. Twelve inches of heading boards are used at both ends of the caps. A square section of the chute set on the footwall is lined with 6-inch by 10-inch by 9-foot material, except when the vertical distance between caps is changed to 5-foot centers. In this case the chute is made of either 6-inch by 10-inch by 10-foot timber or 3-inch by 5-foot lagging of random widths doubled. The manway compartment is lagged over at each floor except for a hole cut in a corner for the ladderway. The parts of the tops of the other compartments not used are also lagged.

If it is foreseen that excessive use will be made of the chute, the raise is driven with four compartments, two of which are used for chutes, one for timber-slide, and one for a manway.

Short raises from the back of a stope to the level above are sometimes driven with only two compartments, in which case the timber slide and manway are placed in one compartment, with the manway on the footwall side under the slide.

Figure 5 shows a type of shrinkage raise driven when it is desired to use the raise as a rock chute. After the raise has been connected with the level above, all broken material is removed from the raise and the raise bottom constructed as shown in the figure. The timber sets in the raise are then removed, beginning at the bottom. After the removal of from one to two sets, broken ore or waste is dropped in the raise from the top, filling it to the bottom of the remaining timbers. Sets are again removed by being hoisted to the top of the raise, and waste or ore is again placed in the chute. This is continued until all timbers have been removed. The work is done with a minimum of danger, as the workmen are under timbers at all times.

In some cases this type of raise is driven with a chute on each end; the manway and timber slide are in the center compartment.

Recently a ventilation raise has been driven from the Hecla 900 level to the No. 3 tunnel for the passage of exhaust air from the Hecla mine.

This raise was driven by contract, the shrinkage method being used. Caps, as shown in Figure 6, were placed at intervals of 5 feet, vertically. The raise was approximately 900 feet in height. For driving of the upper half, a temporary station was cut 450 feet above the bottom of the raise, and a small mine cage was installed to carry men and supplies to the 450-level station. The supplies were then hoisted to the back by a small raise hoist located on the 450-level station.

After the raise was connected with the level above, the broken waste in the chute section was slowly withdrawn; caps were placed in the end of the raise opposite the manway, and the compartment thus made was lined with cedar lagging. Figure 6 shows a cross section of the raise as driven and as completed. Figure 7 shows the bottom arrangement of the raise.

The Star main raise is being driven upward by the shrinkage method, the permanent sets being placed from the top of a completed section as the broken waste is removed.

As the raise is driven, one compartment is timbered by placing two caps across the raise 5 feet apart, center-to-center both horizontally and vertically. This compartment is divided into a manway and a timber slide and is placed directly over one of the center compartments in the completed portion of the raise. The outside of the compartment is lined with a double thickness of 3-inch by 5-foot lagging of random widths, which forms a chute on each end of the raise. A chute gate is maintained at the bottom of a temporary section, and any large bowlders are blasted here before going through a grizzly into the chute in the completed raise.

The raise as driven measures 24 by 9 feet, allowing 1 foot of room around the raise for the blocking of permanent timbers.

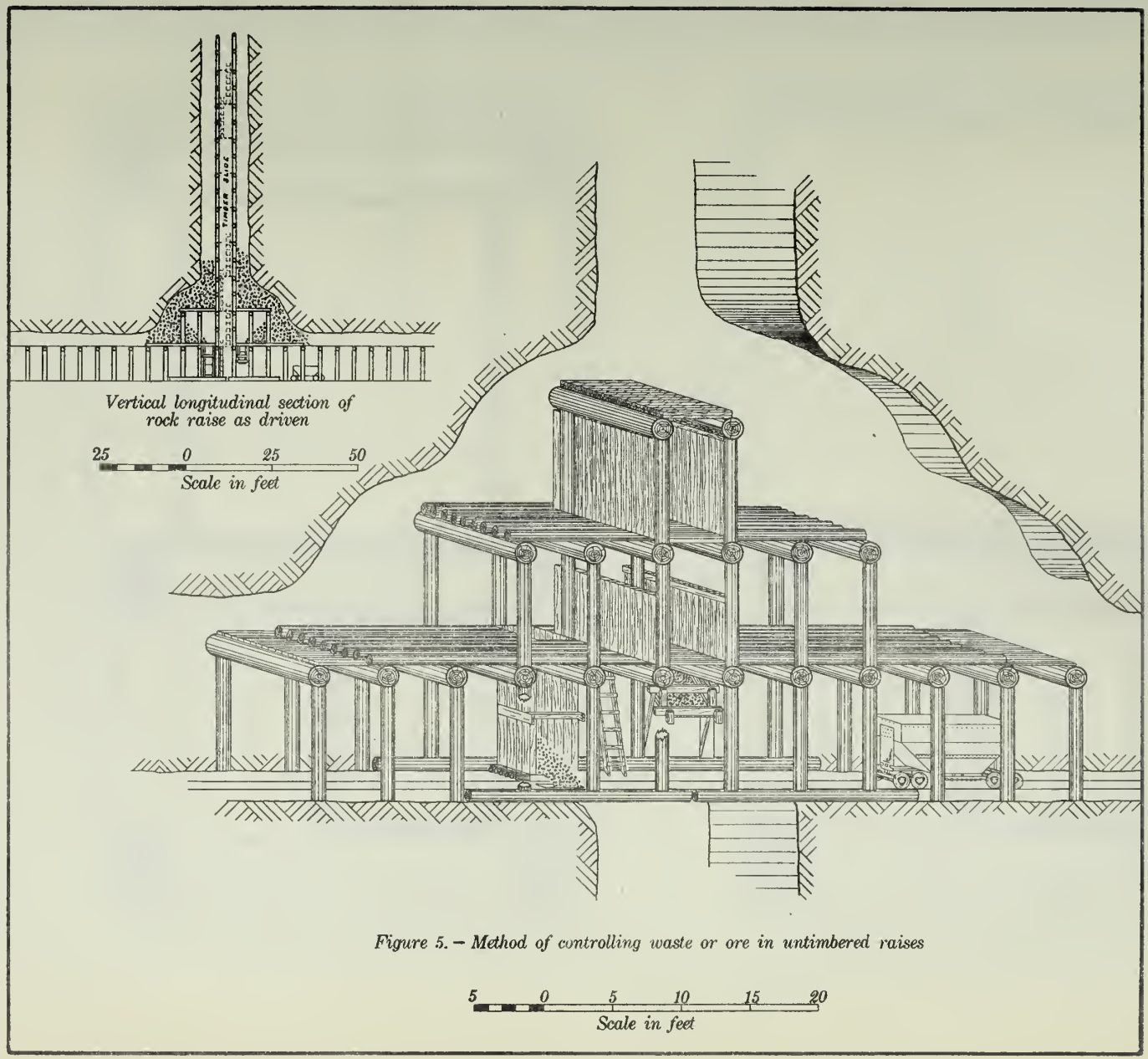
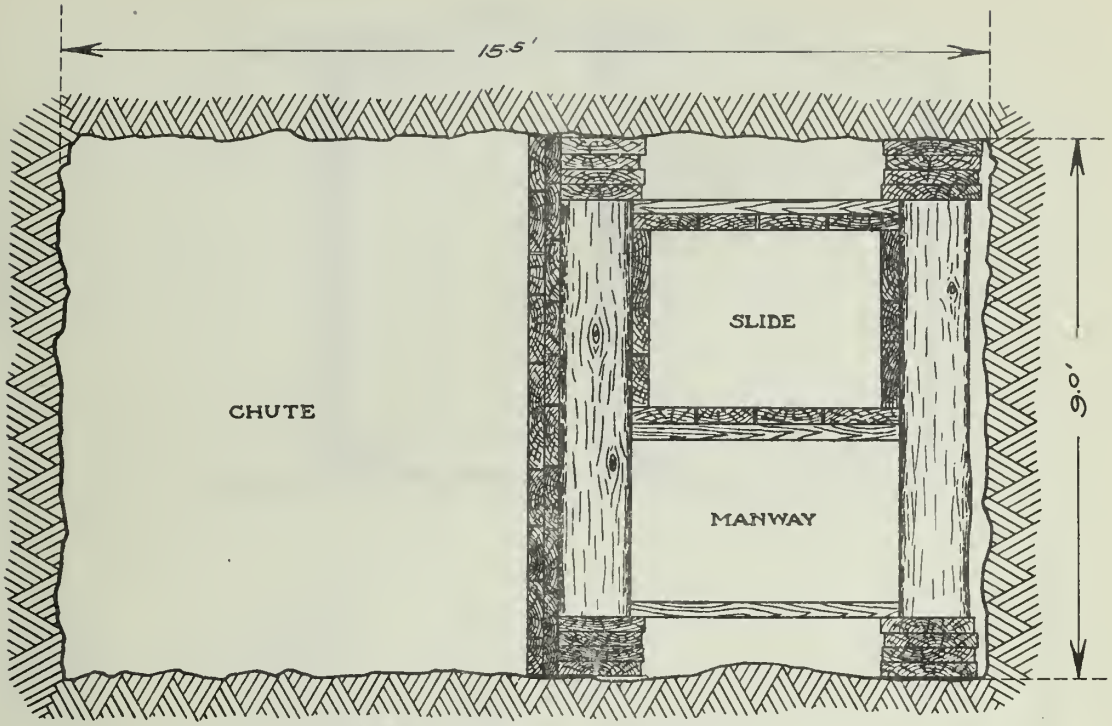
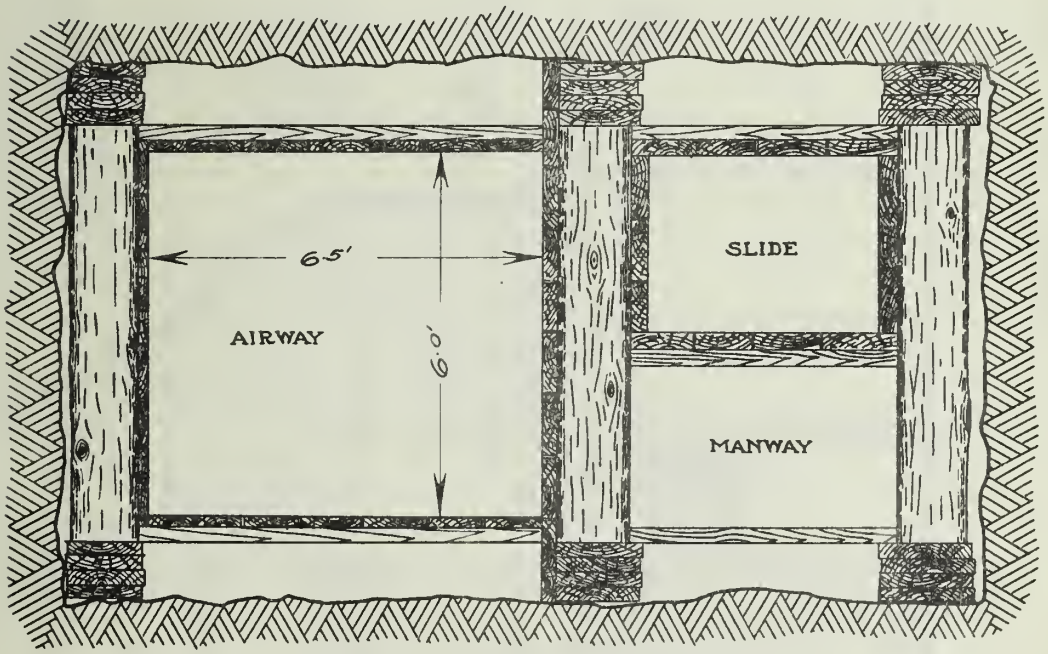


Figure 5. – Method of controlling waste or ore in untimbered raises

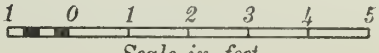


TIMBERING PLAN WHILE DRIVING

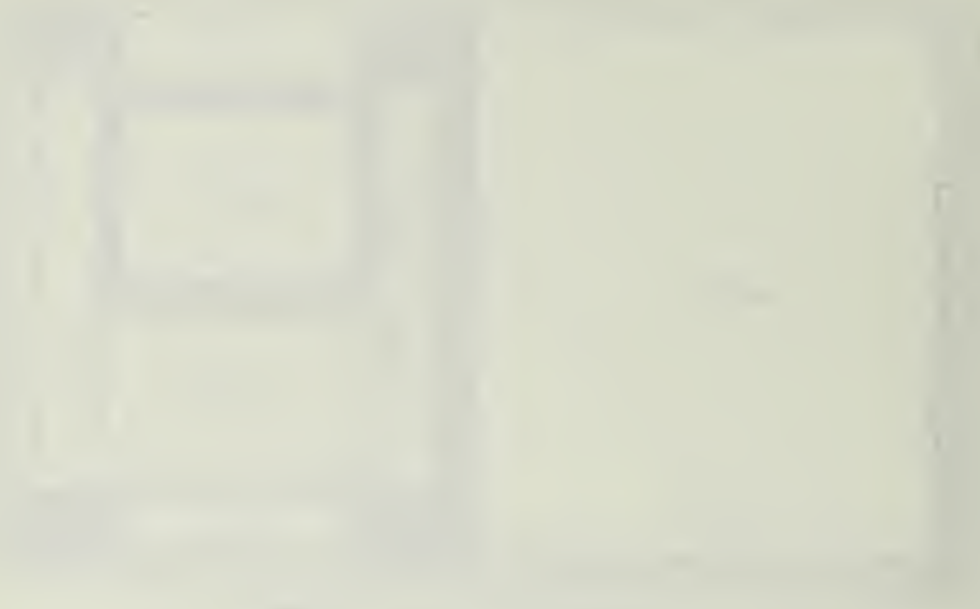


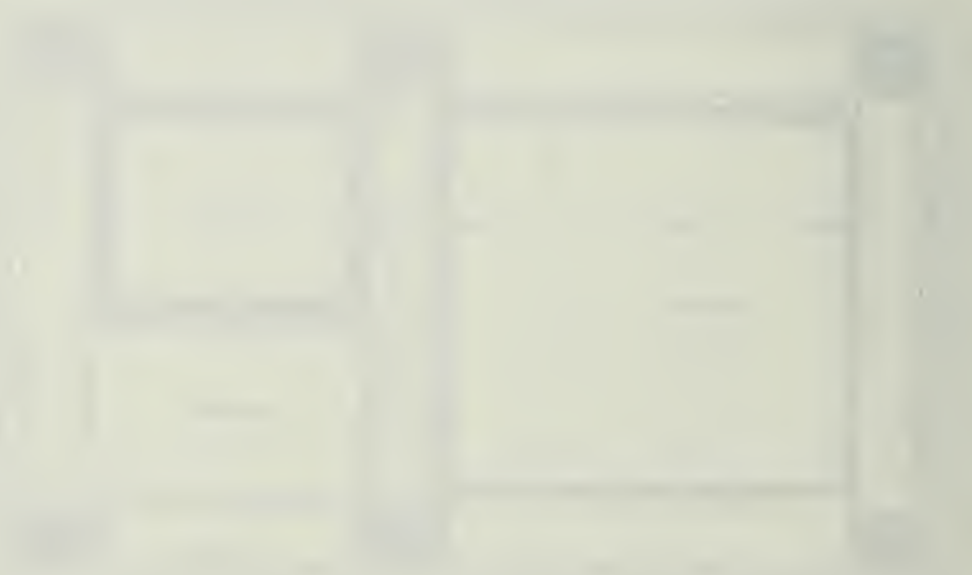
TIMBERING PLAN WHEN COMPLETED

FIGURE 6. – TIMBERING PLANS, VENTILATION RAISE



Scale in feet





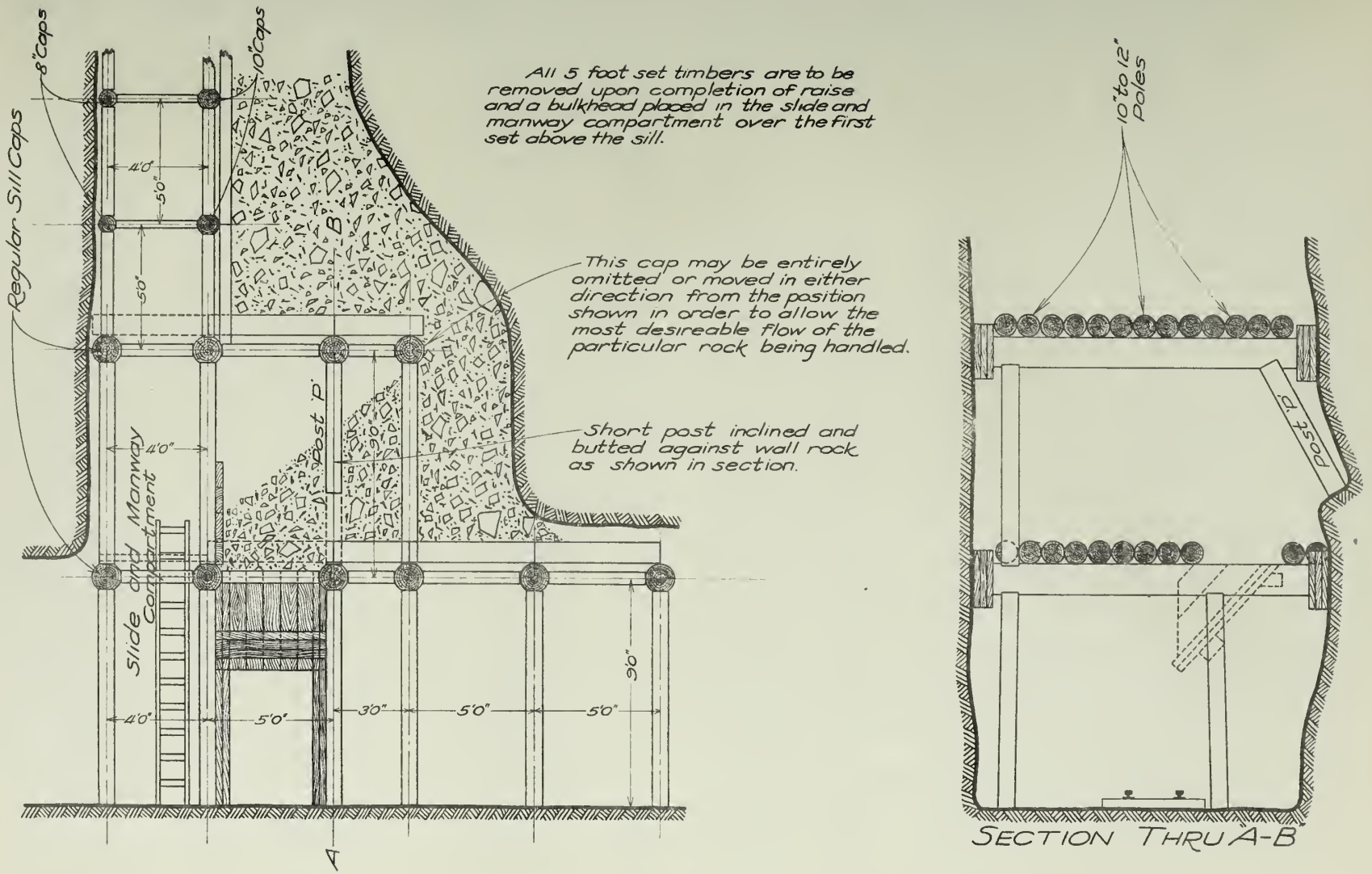


Figure 7.— Bottom of single chute lip shrinkage raise

0 5 10

Scale in feet

A section of the raise is driven to a point about 90 feet above a proposed level. The removal of the temporary timbers and the broken waste is then started, and regular framed shaft timbers are placed, beginning near the top of the raise and leaving room for starting the next lift upward. The permanent timber is added a set at a time, the timbermen working on top of the broken waste, until a connection is made with the timbering of the completed raise below.

As the raise is being driven upward it is checked by the company engineers so that the axis of the raise will be parallel and directly above the axis of the completed raise below. Considerable care is also necessary in placing the first permanent set of timbers at the top of a lift. This method of timbering was considered necessary because of the length of wall plate used, which is 22 feet, and the desire to have the completed raise timbered with unbroken wall plates for the length of the raise.

Lifts of 200, 400, and 600 feet have been driven in this manner. It is the experience in the Star mine that a lift should not be greater than 400 feet.

Drilling is done in raises with wet stopers; 40 holes are drilled to the round. One-inch hollow-hexagonal steel is used, and the starter has the regular cross bit. The remaining steel is fitted with the Carr bit. Blasting is done by electric-delay detonators.

Practically all raises are driven by contract, the price depending upon the type of raise and condition of ground.

The contract for the Star raise includes driving the raise, with temporary timbers, and also the placing of the permanent timbers. Five per cent of the money earned by the contractors is withheld until the permanent timbers are placed, and any expense caused by improper driving of the raise is deducted from the amount due the contractors.

Stoping

Stull Sets and Fill Method.--The general mining method adopted in the earlier days at the Hecla is still used for the production of the larger part of the ore at this mine; the same method is also used for mining the Star ore body. This system may be termed horizontal stoping with stull sets and waste fill.

As stated before, the mining system used in the Hecla main ore body must provide for supporting heavy side and top pressures and allow for mining variable widths of ore.

The stopes in the Hecla have a tendency to widen with depth, causing numerous changes in practice to meet new conditions. The early practice of placing a chute, manway, and timber slide every 50 feet (center-to-center of chute) in stopes in the Hecla vein ore body was abandoned because of excessive side weight, which caused a large amount of repairing in the raises and manways. With this method only about 70 per cent of the excavation was filled with waste. The chute interval was then changed to 100 feet.

A raise, as previously described, is first run to the level above. Beginning at the raise, floors are mined in both directions for a distance of 100 feet. After the first three floors are mined the ore is trammed to the chutes in the end raises. Waste for filling is drawn from the chute in the central raise.

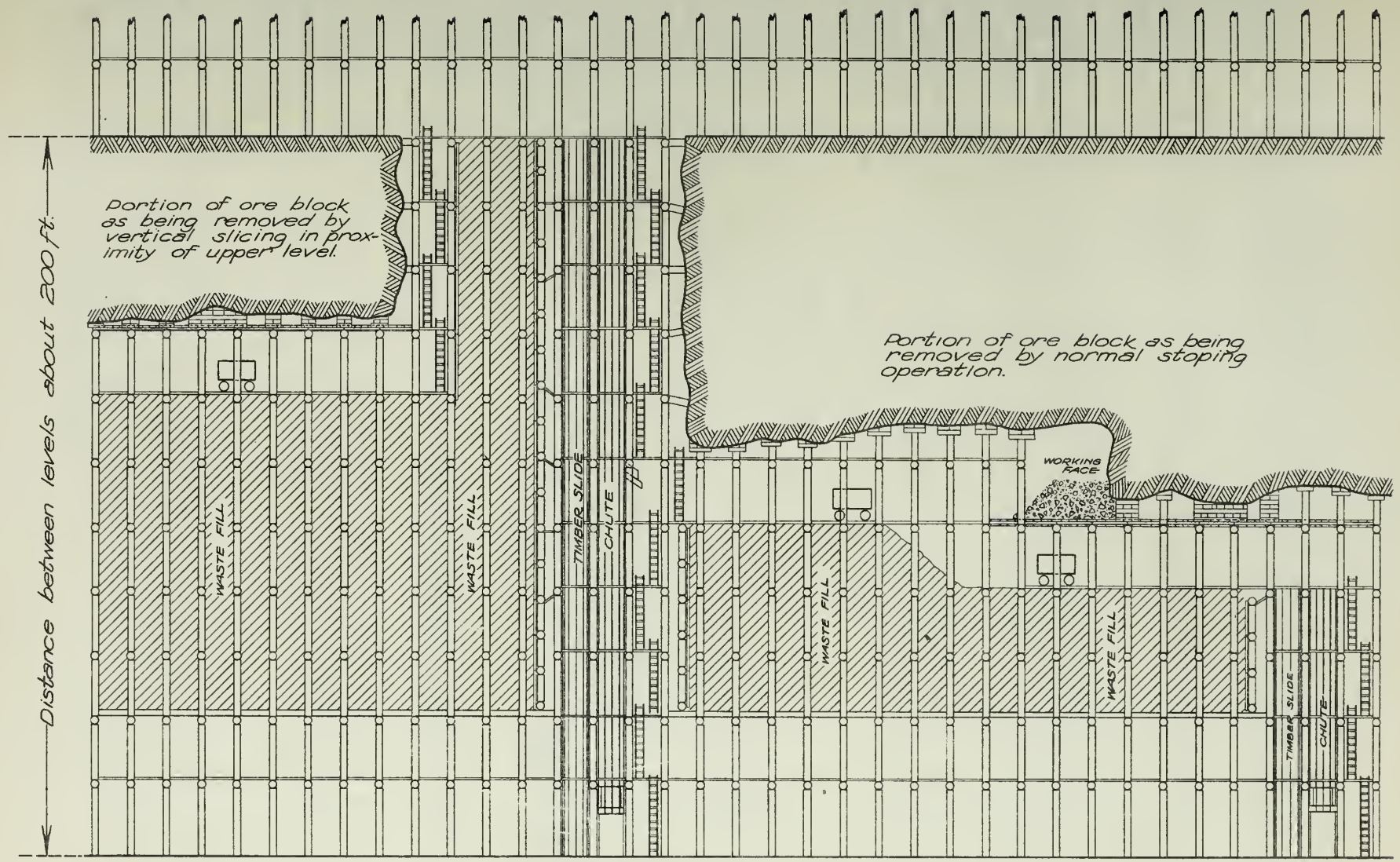
The maximum width of the drift is generally held at 18 feet, but if the ore attains a greater width the stope is widened on the first floor to the full width of the ore. Any ore remaining in the walls of the drift is removed when stoping from the level below the drift.

The first floor is mined from the completed raise to the end of the block and timbered as the face advances. The ore is passed through the top lagging of the drift sets into cars on the haulage level. Before blasting, a lagging is removed from the top of each set (or a floor lagging of the set above) and the space covered by some short cross boards which are pried up one at a time as the cars are loaded. The second and third floors are then mined in the same manner. As the third floor is being mined, the timbers for supporting the filling in the stope can be placed. This bottom of the waste corral is on top of the first floor above the drift; the first floor is kept open to allow repairing the drift timbers or the corral bottom. After the third floor is far enough advanced, the second floor can be filled from the raise, a track being placed on the third floor for the tramming of waste coming from the level above through the completed raise. This waste is taken from development on the level above or from the rock raise connecting with the surface.

When the third floor is completed and the fourth floor is started, this track on the third floor is used for the transfer of broken ore from the face of the stope on the fourth floor, the ore being dropped from the mining floor into the car on the third floor and thence trammed to the chute. When the fourth floor is advanced from 25 to 35 feet from the raise, fill is started on the third floor, coming from coarse waste sorted from mining on the floor above or from the chute connecting with the level above. The track for this waste fill is placed upon the fourth floor.

This method, after the first floor of waste fill has been placed, permits of practically only one floor being open at a time. When wall conditions permit, the track for waste fill is placed on every second floor, and two floors are then open from the top of the filling to the back of the stope.

This general method of advance is carried upward in the stope until only three or four floors, depending upon the ground, of unmined ore remain to the level above. This remaining unmined ore is removed by vertical slicing, starting from the completed raise. As the vertical slice of ore is removed, fill is placed to the level above, between the advancing face and the raise. From 5 to 10 feet is kept open between the face of the vertical stope and the waste fill. When removing back in this manner, stringers are placed on the drift and first floor above to support the drift timbers and the overhead waste gob.



GENERALIZED SKETCH SHOWING METHOD OF STOPE MINING AS PRACTICED AT THE HECLA MINE

0 5 10 20 30 40

FIGURE 8

Note: Girts not shown in sketch.

Figure 8 illustrates the generalized method of mining. The stope to the right of the standard three-compartment raise shows the general advance of floors and the method of filling. The portion to the left of the raise shows vertical slicing to the level above.

Stope rounds consist of vertical rows of three holes each, with one row to each two and one-half feet of face. The rounds are blasted with 30 per cent strength gelatin dynamite.

Round timber is used in the stopes and for drift sets. Figure 4 shows the details of timbering in a stope, with the different sizes of timber and lagging used. Larger timbers are used in the drift and on the first floor than in the stope. The distance from the rail to the bottom of the drift cap is 9 feet to allow for handling 16-foot timbers in the timber slides. As the ground is broken in the stopes and removed, the stull sets are placed from wall to wall. Flooring consisting of 3-inch fir lagging is laid as each set is put in. In extremely bad ground the cap of the set may be placed before the complete removal of broken ore from the face, thus giving immediate support to the walls. The broken ore is used as a staging for placing the caps.

The caps are of variable lengths and are placed from wall to wall of the stope with the necessary vertical posts and horizontal collar braces. The general procedure is to place these caps at 5-foot intervals, measured horizontally, and at 9-foot intervals, measured vertically. As a 16-foot cap is about the maximum in length that can be landed in the stopes, it is sometimes necessary to place two, but rarely three, caps end-on-end to span the stope. In all cases about 12 inches of heading boards are placed against the wall at the end of each cap or string of caps. These heading boards are cut from regular mine lagging 3 inches thick and of random width and are placed parallel to the walls. Any excessive side pressure before the floor is filled is first noticed in the compression of these heading boards, before the failure of the cap. This pressure on the cap is relieved by removing the crushed heading boards and any ground necessary, and placing new heading boards against the original cap.

The timber for the waste corral bottom consists of 6 and 8 inch round lagging cut in 10-foot lengths. Waste timber is placed on top of the round lagging to make the bottom tight.

Caps for the waste or gob fence at the raise are placed 30 inches from the center of the outside raise caps, and spaced 5 feet, center-to-center, vertically; they are blocked to the sides of the stope at each end. Five-foot lagging is placed on the stope side of these gob caps for retaining the filling. Repairs to either the gob caps or the raise caps can be made without one disturbing the other.

This method of mining used is especially adapted to the Hecla mine. It allows for continuous mining and continuous filling, keeping the space between back and fill to a minimum. It allows for the organization of the crew, according to the width of the stope, so that a regular schedule of operation may be maintained. The fact that the waste fill is kept close to the face allows for the use of smaller timbers for caps and posts.

Other Methods.--The East ore body has been mined by various methods. Shrinkage without timbers was not successful in the upper parts of the ore body because of excessive dilution from the hanging wall. The extreme upper portions of the ore body are now being mined by the method used in the Hecla ore body, the waste for fill coming from development above, raises and crosscuts from the stope, and from slabbing of stope walls.

The lower portions are being worked by a timbered shrinkage method. Timbers are placed on 5-foot centers, horizontally, and 9-foot centers, vertically. Floors are removed, with the exception of the lagging immediately over the collar braces, where it is doubled in thickness. Ore is drawn out as in any shrinkage system, the chutes being placed at 15-foot intervals.

Rill stoping, both timbered and untimbered, has been used in parts of the Hecla mine, but it is not adapted to the lower levels, because of increasing widths; its use has therefore been discontinued.

The ore in the Wide West ore body is mined by a straight shrinkage method.

Underground Transportation

An effort is made to standardize all chute lips in the mine so that clearance between mine cars and chute supporting timbers will be sufficient to prevent injury to workmen. Figure 9 shows a standard chute lip.

When it is desired to break the flow of broken material in a chute so that any section may be emptied for repairs, a draw chute (fig. 10) is used.

Broken material is drawn from the chutes into specially designed cars for haulage by a storage-battery motor to the skip pocket. The cars used in the Hecla mine are of 5-ton capacity and have a bottom dump. Figure 11 shows the assembly of this car. Cars used for haulage of Star ore through the Star crosscut are of the same general design but are of 7.5 tons capacity. These cars have a width of 4 feet, 3/4 inches; the car body is 10 feet, 1-1/2 inches long, and the height is the same as the 5-ton car.

Skip pockets are constructed on each level at the shaft. The ore pockets on the 1,600 level and the levels above are rather small, the largest holding about 90 tons. These pockets were cut when mule haulage was the method of transportation.

The skip pockets on the 2,000 level were constructed with the idea of providing storage. An inclined raise was started from the shaft below the 2,000 level and driven to connect with the 2,000 drift. The timbers were then removed from the raise. Another pocket of similar construction was made in the same manner, starting below the first pocket.

Later a station was cut on the 2,000 level, on the opposite side of the shaft from the original station, and a similar skip pocket placed on this side of the shaft. Measuring pockets are not used, the material being drawn directly into the skips. The flow of broken material from the skip pockets is controlled by air-operated gates.

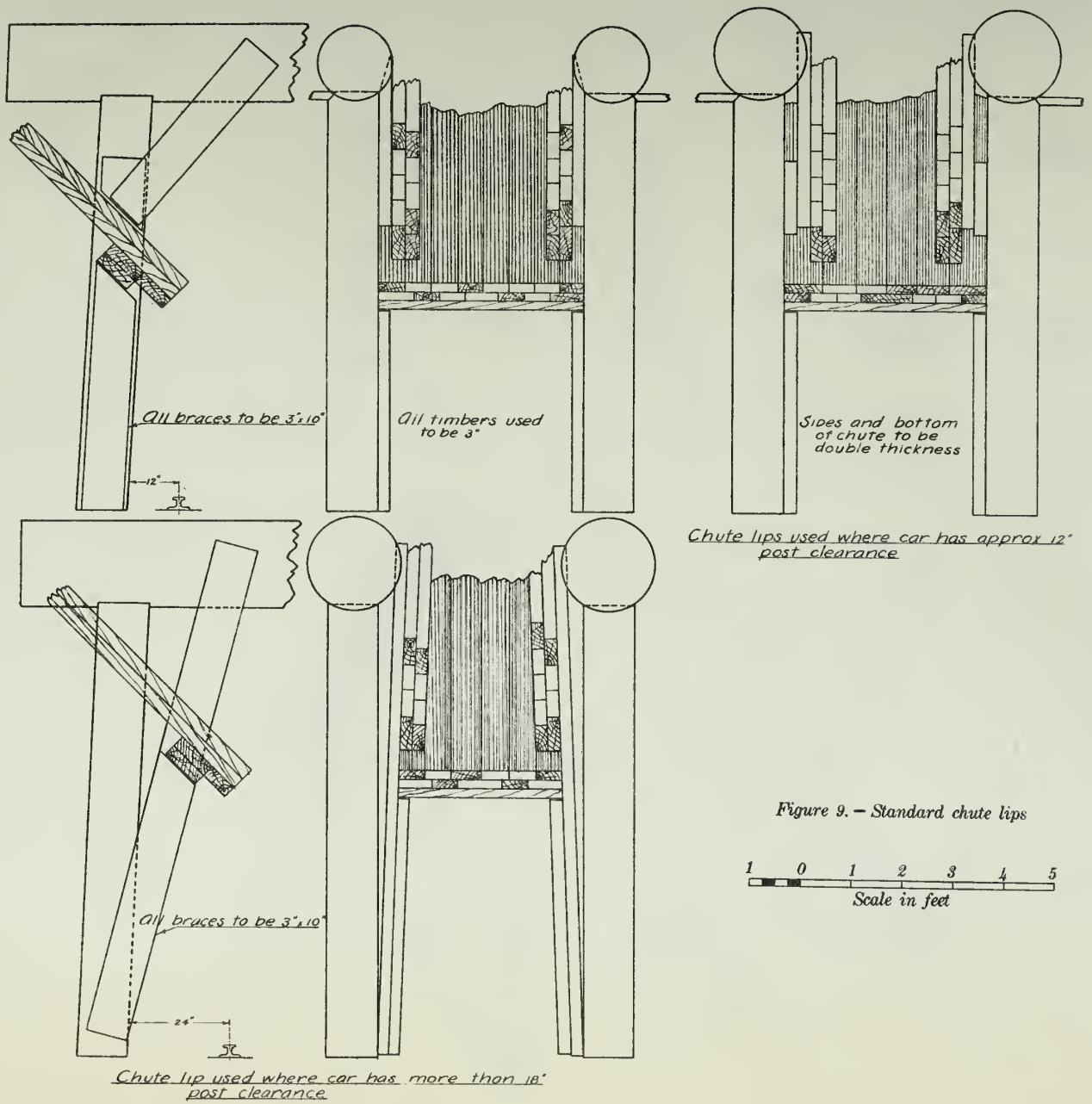


Figure 9. — Standard chute lips

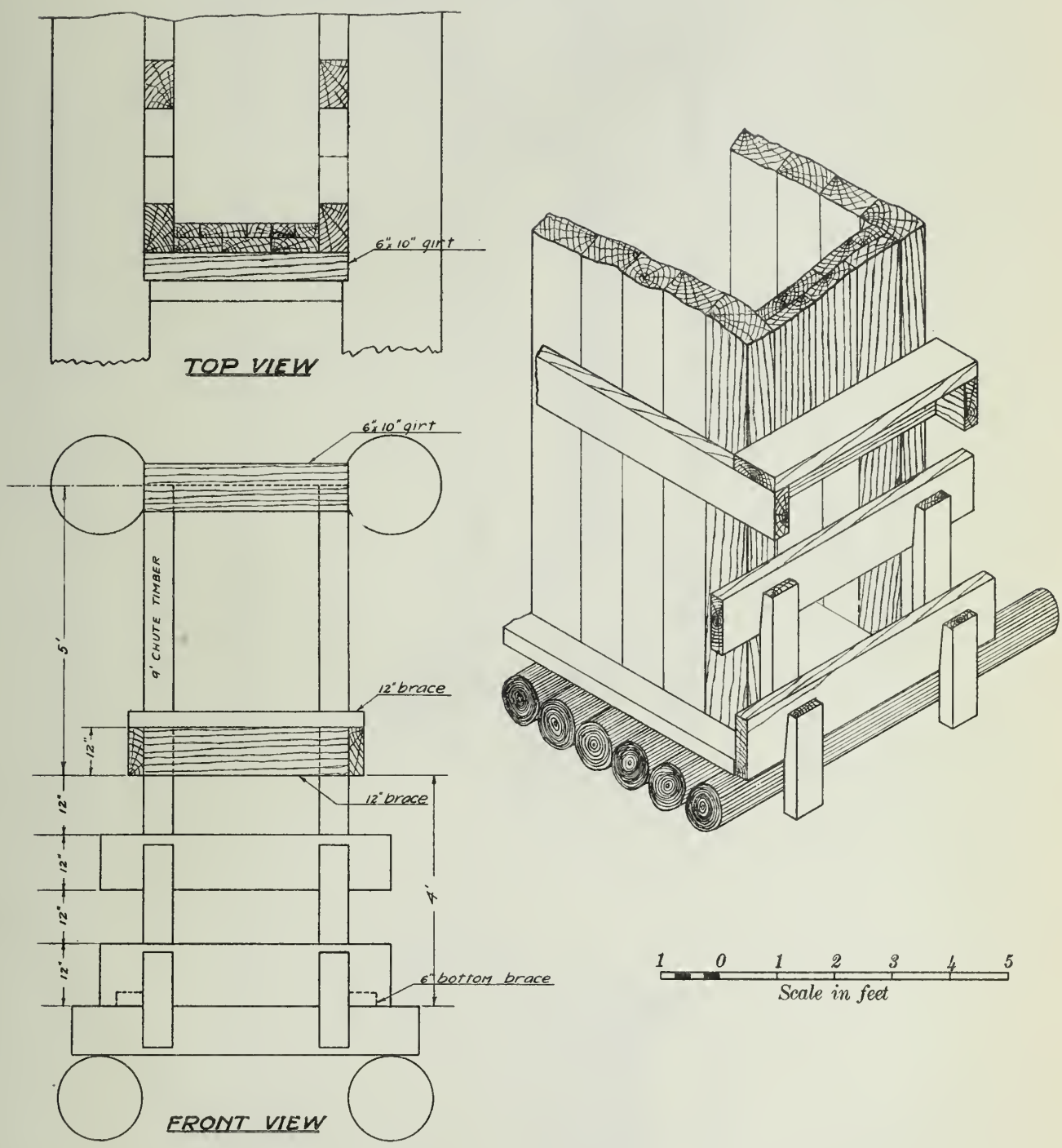


Figure 10. – Standard draw chute

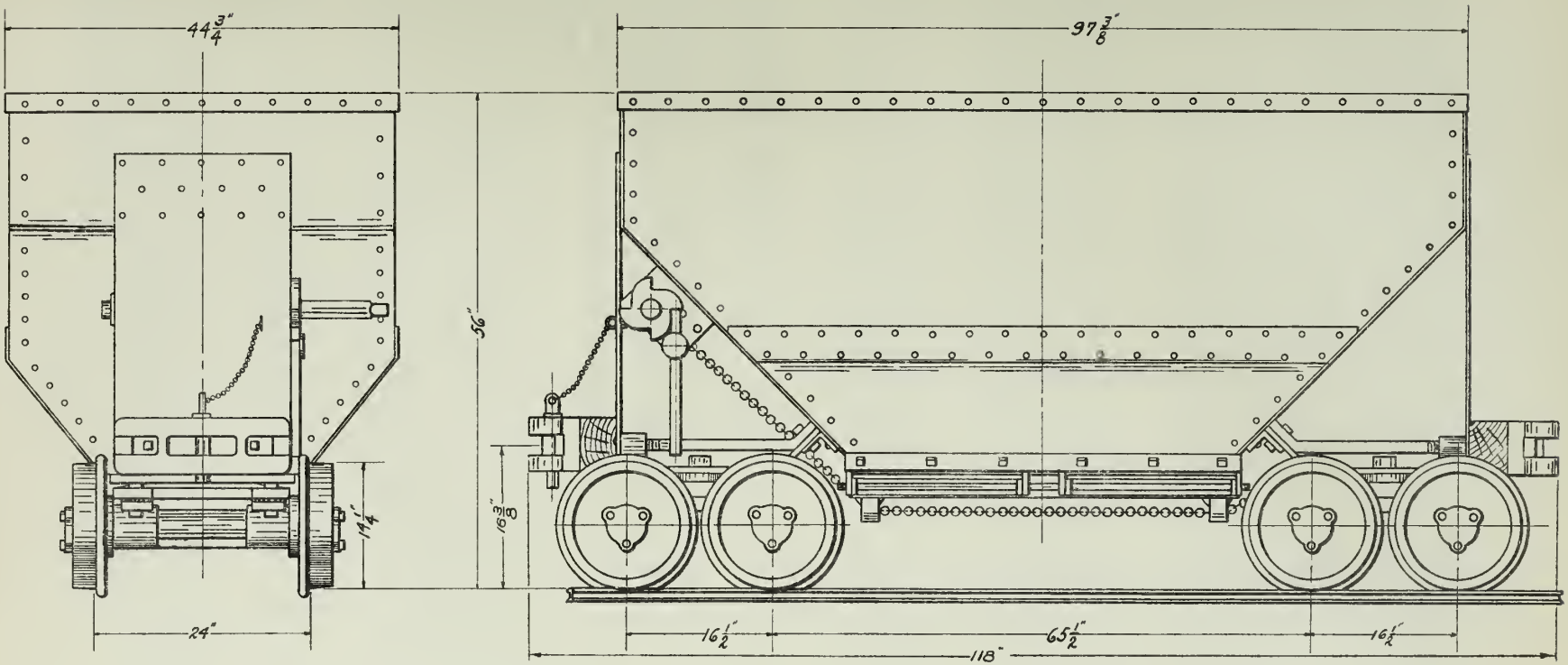


FIGURE 11. – HOPPER-BOTTOM ORE CAR, 80 CU. FT. CAPACITY

0 1 2 3

Scale in feet

Material dropping to the shaft bulkhead under the two ore-hoisting compartments is drawn from the bulkhead into a bucket in the third compartment and hoisted to the 2,000 level where it is dumped into the Hecla ore pocket.

During sinking operations from the 2,000 to the 2,800 level the pocket on the north side of the shaft, originally for Hecla ore, has been used for waste, the broken material from the shaft being dropped into the shaft pocket. The two pockets on the sourth side of the shaft have been used for Hecla and Star ore during this period. The three pockets mentioned have capacities of from 500 to 1,000 tons. Figure 12 is a sketch of the 2,000 station showing the above-mentioned skip pockets.

Three products are hoisted through the Hecla shaft: Hecla ore, Star ore, and waste. These products are taken from their respective pockets in the shaft and hoisted in the skip to bins at the shaft collar; they are diverted to their proper bins by chutes. The position of these chutes is indicated to the hoist engineer by a system of electric signs.

Hoisting of ore and waste was formerly done in a skip of 3-ton capacity. Upon the installation of the new hoisting equipment the capacity of the skip was increased to 7 tons. An auxiliary cage is placed on top of the skip for use of the cager.

Men and material are handled on three-deck cages in both the main hoisting and auxiliary compartments.

A skip-changing device is installed immediately under the floor at the shaft collar to change from skip to cage, or vice versa.

Hoisting equipment installed in November, 1929, and now in successful use, is as follows:

Main Hoist.--The main hoist has a double drum with both drums clutched. The drum diameter is 10 feet; the drum face is 6 feet, running at 76 revolutions per minute to give a maximum rope speed of 2,400 feet per minute. The speed set for hoisting men is 1,300 feet per minute, with full control and safety operation. The hoisting cable is of 1-3/4-inch diameter Lang lay plow steel, being 6 by 19 on the overwind drum and 6 by 21 on the underwind.

The hoist is driven by a 500-volt, direct-current motor of 2,100 horsepower, continuously rated, capable of 4,200 horsepower peak load at a speed of 76 revolutions per minute of the hoist drum.

The motor-generator set driver is a 2,550 kilovolt-ampere synchronous 2,300-volt, three-phase motor, rated at 78 per cent power factor. The direct-current generator produces 1,700 kilowatts at 500 volts. The speed of the set is 514 revolutions per minute.

Material or Auxiliary Hoist.--This hoist has a single drum keyed to the shaft. The drum diameter is 8 feet; the drum face is 6 feet, running at 54 revolutions per minute to give a rope speed of 1,350 feet per minute. The rope is 1-3/8-inch, 6 by 19, Lang lay cast steel. The hoist is driven by a direct-current motor of 850 horsepower, continuous rated, capable of 1,500 horsepower peak load at 54 revolutions per minute of the drum. A 500-volt current is used.

The motor-generator set driver is a 1,050 kilovolt-ampere, synchronous 2,300-volt, three-phase motor. The power factor is 80 per cent. The direct-current generator produces 750 kilowatts at 500 volts, and the speed of the set is 900 revolutions per minute.

Many safety devices are incorporated in the design, being practically in duplicate on the two sets of equipment. These include overspeed, overwind, underwind, power failure, brake wear, excessive acceleration or retardation, clutch and brake interlock devices. Wherever possible, duplicate electrical and mechanical safety devices are provided and the failure of one to operate does not nullify the action of the other.

Control is by means of the Ward-Leonard system, a master controller on the hoist decks operating contactors on a control board. The action of control is completely automatic as to acceleration, retardation, and maximum input, thus preventing any abuse of equipment by the operator.

A complete and detailed description of the above hoisting equipment and factors having influence on the design was given before the American Institute of Mining Engineers at their Wallace meeting on October 5, 1929, by A. C. Stevenson, electrical engineer for the Hecla Mining Co.

EXTRACTION

As a detailed sampling of Hecla stopes is not done, it is not known what per cent of material in place is recovered, or the percentage of dilution. The walls are such that a certain amount of dilution is inevitable.

No effort is made toward close sorting of waste underground; only the larger pieces of waste are removed at this point.

During 1928, 12.44 per cent of the Hecla mine-run ore hoisted was discarded as waste and returned to the mine for fill, and 5.98 per cent was sorted as crude ore and shipped directly to the smelter.

At present, approximately 6 per cent of mine-run from the Star mine is discarded as waste. The ore is of such a nature that sorting out crude ore is not attempted.

WAGE, CONTRACT, AND BONUS SYSTEMS

The employees of the Hecla and Star mines are paid a base wage, plus a bonus which is dependent upon the price of lead for the preceding month. Base wages are as follows:

	Per shift
Shift bosses	$5.25
Timbermen	4.25
Shaft miners	4.75
Cagers	4.00
Motor	3.50
Train loaders	3.25
Miners	3.75
Timber helpers - skilled	3.75
Timber helpers - unskilled	3.50
Shovelers	3.25
Trammers	3.25
Hoist engineers (1st motion)	4.75
Hoist oilers	4.00
Pump engineers	4.00
Compressor engineers	4.00
Blacksmith	4.50
Blacksmith helpers	3.50
Mechanics	4.00 to 5.00
Carpenters	4.50
Laborers	3.00
Ore sorters	3.00

The bonus paid in addition to the base wage is as follows:

Price of lead	Bonus paid per shift
$0.055 - 0.06	$0.25
.06 - .065	.50
.065 - .07	.75
.07 - .075	1.00
.075 - .08	1.25
.08 - .085	1.50
.085 - .09	1.75
.09 - .095	2.00
.095 - .10	2.25
.10 - .105	2.50
.105 - .11	2.75

The price of lead upon which the above scale is based is the quotation of the American Smelting and Refining Co.

All stoping in the Hecla and Star mines is done on company account. Drifts in waste, crosscuts, and raises in ore or waste are generally contracted. No standard price is possible for contracts, due to changing conditions. A contract is given to the entire crew and earnings are divided in proportion to shifts worked. Contract calls for labor only, as explosives and other material are furnished by the company. Raises are also contracted, as previously shown.

VENTILATION

Primary ventilation of the Hecla mine is controlled by a 7 by 3-foot, 6-inch forward-curved blade, primary exhaust, reversible centrifugal fan rated at 95,000 cubic feet of air at 4-5/8 inches water gage. This fan is installed at the surface at the No. 3 tunnel level; it draws air down the main shaft to the 2,000 level and up through the stopes and raises to the tunnel level.

Doors are placed near the shaft on all levels except the 2,000 level to prevent short-circuiting of air currents, sufficient leakage being allowed through each door to give a slight air current in drifts.

It has been found that the old raises through the stopes to the exhaust tunnel were not sufficiently large to permit the proper return of air, and that they were kept open with difficulty. A ventilation raise has been driven from the 900 to the No. 3 tunnel level, and a similar raise will be driven from the 1,600 to the 900 level. This raise and the method of driving is explained under "Raises."

The top, or bottom, as the case may be, of raises is bulkheaded as needed to give proper routing of air through the stopes.

The levels below the 2,000 level are ventilated by an auxiliary fan, having a capacity of 7,000 cubic feet per minute, installed on the 2,000 level at the winze, blowing air through 16-inch ventilation pipe. The return air passes through raises to the main circuit on the 2,000 level.

Other auxiliary ventilation in drifts, crosscuts, and raises is provided by small exhaust fans, either electric-driven or compressed-air driven.

Tests of comfort of working places are made with Kata thermometers.

The main shaft is downcast and rather wet, and considerable trouble was foreseen with ice in the shaft during extremely cold weather. When the surface fan was installed a duct was constructed from the top of the fan to the collar of the shaft and fitted with swinging door regulator to permit the proper amount of warm air to enter the shaft and prevent the formation of ice. This method has been very successful, and temperatures of minus 20° F. have caused no serious difficulty

FIRE HAZARDS

The mines of the Coeur d'Alene mining district maintain a district safety engineer in Wallace, Idaho, the central town of the district. This engineer has headquarters in one of the older type of mine-rescue cars, purchased from the Bureau of Mines. The car is equipped with the necessary oxygen breathing apparatus and accessories and with apparatus for making tests of air and gases.

The engineer in charge of the district makes weekly trips to the larger mines, instructing classes in the use of the apparatus; he is kept thoroughly familiar with conditions at the different working places.

The larger mines have their own apparatus and equipment, which are kept in condition by the district safety engineer.

In case of a mine fire in the district, the car is moved to the mine at once and the combined resources of the mines of the district, which include both men and apparatus, are placed at the disposal of the mine in difficulty. The district engineer takes immediate charge of the situation upon his arrival at the scene of the fire. In an extreme emergency the Bureau of Mines could be called upon for help.

This district instruction in the use of breathing apparatus is supplemented and checked by periodical visits of the regular cars from the Bureau of Mines, when first-aid instructions are also given to the miners of the district.

The surface plant of the Hecla mine is constructed of reenforced concrete, and it is conceded that the transmission of a surface fire to the underground workings is practically impossible.

Underground electrical equipment is placed in concrete structures on the different stations, and the main pumping stations on the 1,200 and 2,000 levels are of reenforced concrete construction.

Fire patrolmen make nightly trips through the mine, punching a watchman's clock with keys placed at strategic points in the mine. These patrolmen report any unusual condition found during the night.

SAFETY

Previous to December 1, 1926, when his services were discontinued, a safety inspector was employed at the Hecla and Star mines.

In December, 1926, a bonus system for shift bosses was started. This system called for the payment of $1 per day for the month in which the shift boss had no accidents causing the loss of one day or more. If the shift boss had less than 1.62 accidents per thousand shifts, he was paid a bonus of 65 cents per day.

This system of bonus had the effect of making a safety inspector out of each shift boss, and during the 25-month period from December, 1926, to January 1, 1929, it caused a reduction of 49.3 per cent in accidents as compared with the time during which the safety inspector was employed.

In January, 1929, an additional bonus system was installed, whereby the crew having no lost-time accidents during a calendar month was a participant in a drawing for two $10 cash prizes. From January 1, to Nov. 1, 1929, a reduction of 53.5 per cent over the number of accidents occurring previous to December, 1926, has been achieved.

When an underground man is hired, he is given a book of rules and general safety suggestions.

An apparatus for introducing a stench (ethyl mercaptan) into the compressed air lines and the ventilating system is installed on the surface.

An emergency exit is maintained from the lower levels to the surface, and raises and levels are marked with "This Way Out" signs. Sketches showing these emergency exits are posted in various parts of the mine, showing the position of the notice and a red line to be followed to the surface.

The Star mine is connected with the Morning mine, and exits both to and from the latter are carefully marked.

Table 1.--Summary of costs, per ton

Name of mine: Hecla mine. Period covered: Year of 1928.

Tons of ore handled: 312,942.

Not segregated as to method of mining.

	Labor	Super-vision	Comp. air, drills and steel	Power	Explo-sives	Timber	Other sup-plies	Total
Development in ore	.120	.011	- -	- -	- -	- -	.079	.210
Mining	1.030	.030	- -	- -	- -	- -	.101	1.211
Transportation	.220	- -	- -	.027	- -	- -	.075	.322
General	.191	- -	- -	.036	- -	- -	- -	.227
Surface expense	.216	- -	- -	1/.082	- -	- -	.062	.360
Not segregated	- -	- -	.070	- -	.112	.357	- -	.539
	1.777	.091	.070	.145	.112	.357	.317	2.869
Development in waste	(2)	(2)	(2)	(2)	(2)	(2)	(2)	.428
								3.297

1 - Includes compressors.
2 - Not segregated.

Table 2.--Summary of costs in units
(Based on tons hoisted)

Name of mine: Hecla mine Period covered: Year, 1928.

Tons handled: 312,942.

	Miners	Shovelers and stope tramners	Timbermen and helpers	Total direct labor
A. Underground labor:				
Man hours per ton:				
Horizontal stoping				
(Hecla ore body)	.256	.451	.611	1.318
Development in ore				
(Hecla ore body)	.351	.149	.290	.790
1/Combined methods				
(East ore body)	.389	.220	.785	1.394
2/Shrinkage methods				
(Wide West)	.886	.274	.250	1.410
Average for mine	.313	.343	.606	1.262
General labor (level trainmen, timber handlers, and hoistmen)	- -	- -	- -	.381
Repair crews	- -	- -	- -	.326
Supervision (underground shift bosses only)	- -	- -	- -	.061
Total man hours per ton	- -	- -	- -	2.029
Tons mined per shift:	- -	- -	- -	3.9

B. Power and supplies:

Explosives (lbs. per ton):		
Horizontal stoping (Hecla ore body)		.533
Development in ore		.947
1/Combined methods (East ore body)		1.174
2/Shrinkage (Wide West ore body)		2.054
Average		.783
Timbers -- not segregated as to mining method:		
Stulls (linear ft. per ton)		1.234
Sawed timber (bd. ft. per ton)		11.207
Power -- not segregated as to mining method (kw.h.):		
Compressors	3,432,330	
Hoist	1,362,197	
Ventilation	263,470	
Lighting	267,250	

1/ Includes timbered shrinkage, horizontal stoping (as in Hecla ore body), and development in ore. Figures based on tons hoisted.

2/ No development work. Figures based on tons drawn from chutes. If based on tons broken, would be: Miners .651; shovelers .201; timbermen .184; total 1.04 man hours per ton. Powder consumption would be 1.509 lbs. per ton.

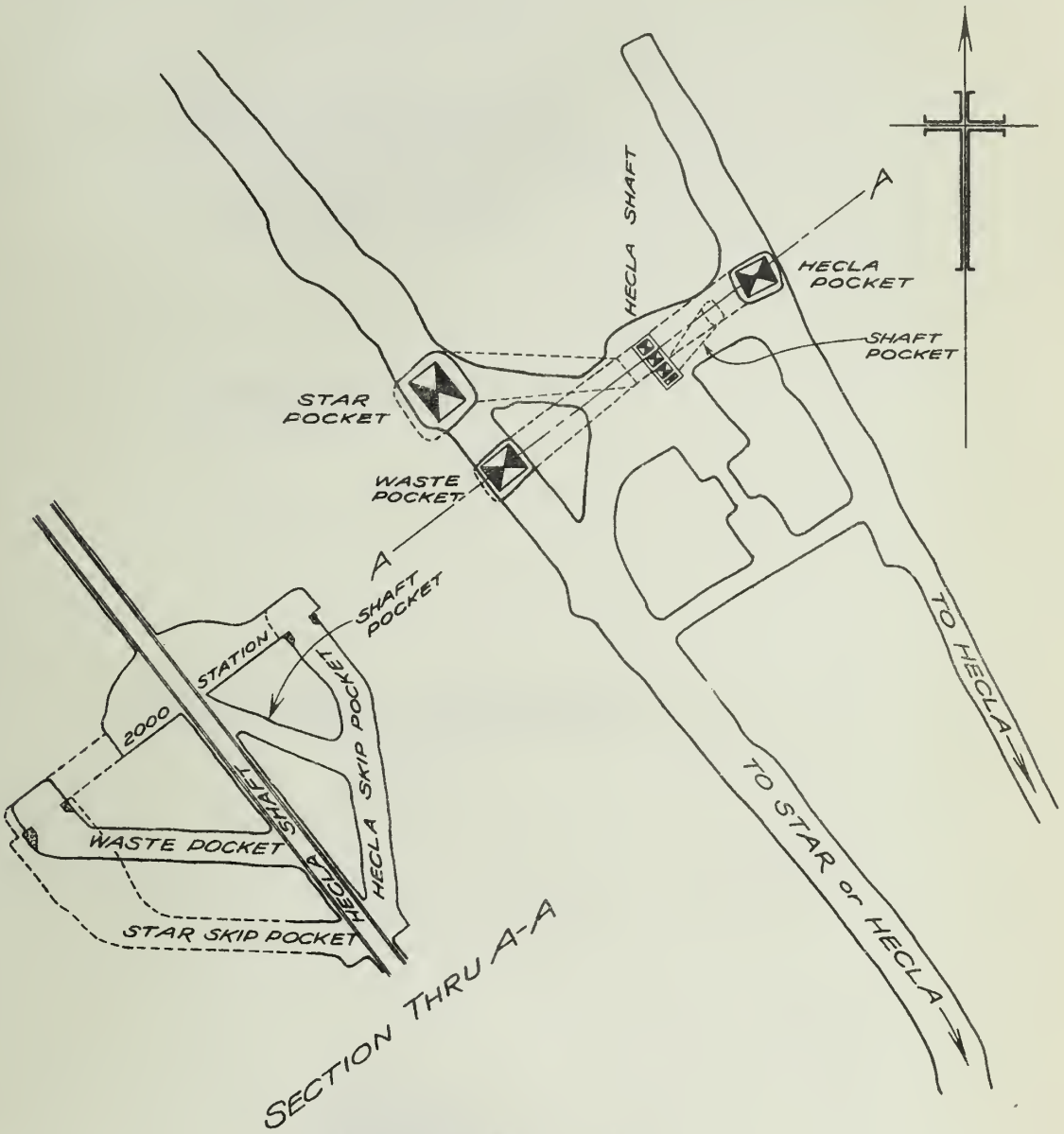


FIGURE 12.-

ARRANGEMENT OF SKIP POCKETS ON 2000 LEVEL OF THE HECLA FOR HANDLING ORE AND WASTE FROM STAR AND HECLA MINES.

U. of I. DUPS.
1942

I. C. 6233

JANUARY, 1930

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

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GEOPHYSICAL ABSTRACTS

NO. IX

BY

FREDERICK W. LEE

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GEOPHYSICAL ABSTRACTS[1]

No. 9.

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1. The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6233."
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1. GRAVITATIONAL METHODS

DIE REGISTRIERUNG DER SCHWINGUNGEN EINES SCHWERKRAFTPENDELS MITTELS HOCHFREQUENTER ELEKTRISCHER SCHWINGUNGEN UND IHRE RÜCKWIRKUNG AUF DAS PENDEL

(REGISTRATION OF OSCILLATIONS OF A FORCE OF GRAVITY PENDULUM BY MEANS OF HIGH-FREQUENCY ELECTRIC OSCILLATIONS AND THEIR REACTION UPON THE PENDULUM)

By H. Schmehl

Zeitschrift fuer Geophysik, vol. 5, No. 2, 1929, pp. 53-58.

In 1928 the author made observations of the reaction of pendulum registration to high-frequency electrical oscillations. In order to obtain more precise data Schmehl continued these observations in 1929 after some improvements of the instruments were made.

The effect of the new registration arrangement, which works by means of high-frequency electric oscillations as well as by capacity constants, is examined.

The gilt, nickel-steel, half-second pendulum, constructed by M. Fechner is described and a table showing the observations made with this pendulum is given.

After examination of the results of the pendulum observations the author concludes that the registration of pendulums by means of high-frequency currents may be applied with good results in making relative measurements of the forces of gravity.--W. Ayvazoglou.

DIE REGISTRIERUNG VON PENDELSCHWINGUNGEN MITTELS KAPAZITIVER KONSTANTE

(REGISTRATION OF PENDULUM OSCILLATIONS BY MEANS OF CAPACITY CONSTANTS)

By H. Mahnkopff

Zeitschrift fuer Geophysik, vol. 5, No. 2, 1929, pp. 49-53.

The article deals with experiments performed partly by P. Lejay in Zikawei (China) and Paris and partly at the Geodetic Institute in Potsdam.

Experiments have proved that the electric contacts of a precision pendulum-clock can be replaced by comparatively simple devices working by means of high-frequency short-wave oscillations, and that the oscillations of the

force of gravity pendulums can be registered with a high accuracy with the aid of capacity coupling.--Author's Abstract translated by W. Ayvazoglou.

INTERRELATION BETWEEN THE LOCAL GRAVITY ANOMALIES AND THE DERIVATIVES OF THE POTENTIAL

By B. Numerov

Zeitschrift fuer Geophysik, vol. 5, No. 2, 1929, pp. 58-62.

Numerov derives formulas by which the potential and its derivatives may be computed if the distribution of gravity over a plane, presented by a part of the earth's surface, is known. Thus, without taking into consideration the remotest zones and without extending the integration over the whole globe, the differences between the derivatives of the potential for contiguous points may be found.

Of practical value are the first derivatives of the potential of disturbing masses, which show the deflection of the plumb line U_x and U_y along the x axis and the y axis.

The possibility of computing the deflection of the plumb line on the basis of gravity anomalies may be of great practical importance, especially in remote regions where the topographical surveys are to be based on astronomical observations instead of triangulation.--W. Ayvazoglou.

UEBER DEN GRÖSSTEN MÖGLICHEN SCHWERE-UNTERSCHIED

(ON THE HIGHEST POSSIBLE DIFFERENCE OF THE FORCE OF GRAVITY)

By Karl Jung

Zeitschrift fuer Geophysik, vol. 5, No. 2, 1929, pp. 65-72.

The author shows how to estimate the value of the highest possible difference of the force of gravity between two stations caused by the density differences in the subsoil. The estimation is considered to be adequate for all cases.

The results are given in the form of curves suitable for use in practice.

The author concludes the article by a brief discussion of the results. He says: "Examination of the curves shows us that $\frac{\triangle g}{\triangle \delta}$ increases with the increase of a (half distance between the stations) from 0 to the highest value. This highest value is equal to about 40 t (t in kilometers). Of course this estimation is valid for great distances between the stations only. Thus, where a = t, about three-fourths of the highest value $\frac{\triangle g}{\triangle \delta} = 40\ t$ is reaches; where a = 1/10 t, about one-fifth; where a = 1/100 t, about one-twentieth; and where a = 1/1000 t, about one-hundredth.

Differences of the force of gravity caused by density differences can not be higher, and if observed may result from the inaccuracy of measurements or corrections, provided that no assumptions on the distribution of densities in the subsoil which differ greatly from those known so far are made."--W. Ayvazoglou.

ERHÖHUNG DER BEOBACHTUNGSGENAUIGKEIT BEI RELATIVEN PENDELMESSUNGEN

(INCREASE OF ACCURACY OF OBSERVATIONS WITH RELATIVE PENDULUM MEASUREMENTS)

By O. Meissner and H. Martin

Zeitschrift fuer Geophysik, vol. 5, No. 3-4, 1929, pp. 145-151.

Two simple photographic methods of registration for determining differences in time of oscillation of various pendulums are given: (1) The registration of pendulum oscillation, and (2) the photographic coincidence proceeding.

The first method is described by O. Meissner, who is of the opinion that the present time accuracy of gravity measurements with relative pendulum measurements, which amounts to about 1×10^{-3} cm. sec. $^{-2}$, is in many cases not sufficient for geophysical exploration work and therefore must be increased up to 2×10^{-4} cm. sec. $^{-2}$. O. Meissner describes his method, which is based on the improvement of the method of observation of differences in the time of oscillation.

The experimental arrangement, as well as the interpretation of the data obtained, is explained.

The second method, described by H. Martin, consists of improving the accuracy of observation of coincidences. The arrangement of the experiment explaining this method is given in a schematical design, and a table of a series of measurements shows that the desired accuracy can really be obtained.--W. Ayvazoglou.

BASES CIENTIFICAS Y PRACTICAS DEL METODO GRAVIMETRICO

(SCIENTIFIC AND PRACTICAL BASES OF THE GRAVIMETRIC METHOD)

By W. Olhovitch

For his thesis Olhovitch was granted the diploma of a Mexican petroleum engineer by the National University of Mexico, in November, 1929. According to the statement of Juan Korzujin, Professor of the Mexican National University, the pamphlet is of real interest. He writes: "Geophysicists will agree that until now there has not existed a full description of gravitational surveying, although the literature on this subject is quite extensive and the theory, as well as the practical use of the gravitational balance, must be considered to be widespread. There are many booklets and articles containing full

information on the torsion balance, but it is noticeable that until the present there has not been even a fixed denomination of the coefficients.

"It is also to be mentioned that many authors of short treaties on the torsion balance too often omit the mathematical development of their formulas, which makes a thorough study of the theory of the torsion balance unnecessarily difficult.

"The main task of Olhovitch's work consists of issuing a concise summary and unification of different systems by which the gravitational balance is explained and of deriving formulas by which the understanding can be facilitated. From my practice in teaching geophysics I know how much time may be lost by a student in deriving such formulas, as for instance formula 6 (p. 9) or Figure 4 (p. 23) of L. Oertling's booklet, without speaking of the far more scientific works of P. Nikiforov and many others. Pure mathematical problems should not be left unsolved in a book dealing with geophysics. In Olhovitch's book all these unnecessary difficulties are eliminated, thus it certainly will be much appreciated by all young student geophysicists. The examining body of professors decided to have the book published by the National University of Mexico. The book is written in Spanish, but its publication in English might be of great value."--Juan Korzujin.

CONTRIBUTION TO THE QUESTION OF THE DISPLACEMENT OF THE ZERO-POINT OF THE TORSION BALANCE IN A GRAVITY VARIOMETER

(In Russian)

By P. M. Nikiforov, S. K. Ghirin, and D. G. Uspenski

Bulletin of the Institute of Practical Geophysics,
Leningrad, No. 4, 1928, pp. 268-292.

The authors give some of the results of their experiments carried out in connection with the displacement of the zero-point of a torsion balance caused by the reaction of the wire to heating and cooling, as well as by the effect of convection currents on the balance beams. The instruments used for experiments were the variometers constructed by the Physical Institute in Leningrad (Nos. 101 and 103) and the Schweydar-Bamberg instrument No. 87.

Several figures given in the article show the behavior of the beams at different temperatures.

A connection between the displacements of the zero-point and variations in temperature has been established with certainty by the experiments. At the same time the existence of a great complexity in this dependence has been found.

In the opinion of the authors the following conclusions may be drawn from the experiments:

1. Temperature fluctuations in the surrounding air strongly affect the position of the zero-point of the torsion balance in a gravity variometer.

2. The displacement of the zero-point of a torsion balance is closely connected with the effect of convection currents on the beam within the protecting casing.

3. The behavior of both beams of a gravity variometer may be different under equal conditions of temperature fluctuations.

4. In case the temperature of the instrument is not steady the position of the zero-point of the beam can not be determined as a function of the temperature only, therefore a correction of the temperature coefficient in a general way is not sufficient.

5. For comparatively large values of the temperature gradient relative to time, of the order of 2° per hour and more, the position of the zero-point is not a function of this gradient only, in some cases one and the same value of the gradient may cause deflections of the beam different in magnitude, as well as in sign.

6. For small values of the temperature gradient relative to time, of the order of 1° per hour and lower, the undulatory portion of the curve disappears, thus a strict connection between the position of the zero-point and the value of the temperature gradient is evident (but not between the position of the zero-point and the temperature itself).

7. The magnitude and the signs of the displacements of the zero-point of a torsion balance depend upon the position of the beam and loads with relation to the walls of the protecting casing--that is, with relation to the ascending and descending convection currents.

8. The displacement of the zero-point depends to a high degree upon the position of the lower load with regard to the bottom and the top of the lower tube of the instrument. The authors connect this phenomenon with the change of directions of convection currents at these points of the tube.

9. The occurrence of convection currents is conditioned by the existence of the temperature gradient in the direction of the radii from the axes of symmetry to the walls of the instrument.

10. Displacements of the beam in the same direction may correspond to horizontal and vertical temperature gradients having different signs, and inversely, deflections of the beams having different signs may correspond to temperature gradients having the same sign.

11. During the heating and cooling of the instrument the sign of the displacement of the beam depends at the same time upon the height of suspension of the lower load and upon the angle between the beam and the plane of symmetry of the protecting casing.

12. The deflection of the zero-point can not be explained by the effect upon the beam of the flow of momentums the origin of which is the increase of particle energy during the rising of the temperature.

13. Study of questions concerning the investigation of convection currents inside of the envelope of the instrument, as well as of how to direct these currents so that their effect upon the displacement of the beam may be a minimal one, is very important.

14. Protecting the instrument against variations of external temperature is important in obtaining accurate readings.

15. The reduction of the effect of convection currents upon the displacement of the zero-point of a torsion balance to the minimum will certainly be possible by placing the balance in a vacuum. This is a problem of technical construction.

16. The results of the authors' experiments point out the importance of working in the field with continual recording of the oscillations of the beam of a gravity variometer.--W. Ayvazoglou.

GRAVIMETRICAL PROSPECTING OF FERRIFEROUS QUARTZITES IN THE DISTRICT OF KRIVOY ROG

(In Russian)

By P. M. Nikiforov, S. K. Ghirin, and D. G. Uspensky

Bulletin of the Institute of Practical Geophysics, Leningrad, No. 4, 1928, pp. 315-326.

The authors give the results of the researches made in 1927 in addition to those made in 1925 and 1926 and already described in a previous work (Lee, F. W., Geophysical Abstracts, No. 1. "Gravimetric researches of ferruginous quartzites in the region of Krivoy Rog." I. C. 6120, Bureau of Mines, 1929, 25 pp.)

For the survey, the whole area was divided into two sections, the northern and the southern. In order to obtain more detailed results the authors adopted the system of prospecting lines 100 meters apart. Six torsion balances have been used--four constructed by the Physico-Mathematical Institute in Leningrad, one Hecker's instrument type 1913, and one Schweydar's type 1926.

The following table shows the comparative efficiency of the different instruments.

Comparative Efficiency of Six Torsion Balances

INSTRUMENT	Number of observations	Number of stations	Number of working days	Number observations per day	Number of stations taken per day
Physico-Math. Inst. No. 101	154	116	73	2.11	1.59
Do. No. 102	221	197	77	2.74	2.56
Do. No. 103	210	172	73	2.88	2.36
Do. No. 104	225	201	68	3.31	2.96
Type Hecker (1913)	146	116	74	1.97	1.57
Type Bamberg No. 87	72	62	44	1.64	1.41

New pavilions 1 by 1 by 2 meters in size have been used for protecting the instruments against the change in meteorological conditions.

A geological map based upon the survey is added. A comparison of gravimetrical prospecting with the data obtained by drilling is shown in the corresponding two maps. Although the correctness of the results obtained by gravimetrical prospecting is well proved by the comparison of them with the data obtained by drilling, the authors are of the opinion that a systematical control by drilling is necessary.--W. Ayvazoglou.

ON A RELATION BETWEEN THE DISTRIBUTIONS OF GRAVITATIONAL ANOMALIES AND THE ORIGINS OF EARTHQUAKES IN JAPAN

By Chüji Tsuboi

Proceedings of the Imperial Academy, Tokyo,
vol. 5, No. 8, October, 1929, pp. 326 - 330.

Tsuboi investigates the possible relation between the distributions of the gravitational anomalies and the origins of earthquakes in Japan. An actual comparison of these two distributions has shown a remarkably close correlation existing between them.

Measurements of the intensity of gravity, at not lessthan 122 stations, have been completed over the main part of Japan. For each station the gravitational anomalies $g_0'' - \gamma_0$ were calculated, where g_0'' is the intensity of gravity corrected for the free-air, terrane, and Bouguer effects, and γ_0 is the intensity of gravity as calculated by the Helmert formula. The isostatic reductions were not made.

Two maps showing the distribution of the centers of the earthquakes are given. The earthquakes taken into consideration were those which occurred within the last 15 years, amounting to about 1000 in number.

By comparing the two maps, Tsuboi established that the gravitationally anomalous regions were also regions most frequented by earthquakes; that is, that earthquakes occur in gravitationally anomalous regions more frequently than in others.

Tsuboi proposes four alternative explanations: (1) That the gravitational anomaly causes earthquakes, (2) that the earthquakes cause the gravitational anomaly, (3) that gravitational anomaly and occurrence of earthquakes are produced by one common cause, (4) (with the least probability) that the gravitational anomalies and earthquakes are entirely independent phenomena and the apparent coincidence of the distributions of them is merely accidental.

He concludes that a decisive choice of any one of these alternatives should be made after a more complete accumulation of data is obtained.--W. Ayvazoglou.

2. MAGNETIC METHODS

A NEW MICROMAGNETOMETER

By Frank Rieber

American Institute of Mining and Metallurgical Engineers, Geophysical Prospecting, 1929, pp. 401-415.

After a brief description of the relation of simpler structures to the magnetic anomalies caused by them and of an approximate estimation of the accuracy with which these anomalies can be measured with the instruments now in use, Rieber describes a new device by which the degree of accuracy and dependability of magnetic instruments may be increased considerably. The experimental model of the new instrument had shown an apparent accuracy of over 30 times that of typical field balances. As it was realized that this instrument must first of all maintain the vertical and horizontal component measurements in an absolutely vertical and horizontal direction, a heavier form of tripod has been adopted, together with the level bubble tubes of the type employed by the Coast and Geodetic Survey.

These levels can be read to 2 seconds of an arc, or approximately 0.0005°. The arrangements made for intensifying the sensitivity of the new instrument are explained and shown in accompanying figures.

Photographic pictures of the instrument set up for visual tests and for photographic test are given. During the experiments the effects of the length and dimensions of the vibrating wire and the arrangement of the magnetic bars were given attention, with the result that a few small changes were made and a second model is now under construction.

The author concludes: "Due to the lightness and the strength of the wire which constitutes the one moving part, and to other features of construction, greater dependability as well as greater accuracy may be hoped for."--W. Ayvazoglou.

I.C.6233.

EIN TRANSPORTABLER ELEKTRISCHER MAGNETOMETER

(A TRANSPORTABLE ELECTRIC MAGNETOMETER)

By W. Uljanin

Terrestrial Magnetism, vol. 34, No. 3, Sept., 1929, pp. 199-207.

An electrical method for determination of the horizontal intensity of the earth's magnetism is described. The magnetometer has been designed by the author and manufactured by the firm of Prof. Dr. Edelmann und Sohn in Munich.

The magnetometer has been carefully examined and proved to be of practical value.

A detailed description of the magnetometer and of the potentiometer is given and a scheme of the system of connection shown.

The method for determining the constant of the magnetometer is explained. In the last section of the article Uljanin gives his experience with different types of Weston cells, as in his opinion, the electrical method of measuring the horizontal component is based on the exact knowledge of the electromotive force of the applied standard cell.--W. Ayvazoglou.

A DEMONSTRATION OF THE REFLECTION OF GEOLOGIC CONDITIONS IN OBSERVED MAGNETIC INTENSITY

By H. R. Aldrich

American Institute of Mining and Metallurgical Engineers, Geophysical Prospecting, 1929, pp. 385-400.

The paper presents a diagram of dip-needle readings across the Gogebic iron range and the Keweenawan formations in part of northern Wisconsin. The diagram is studied in connection with the geological map and serves to demonstrate the extent to which such methods may aid in solving geological problems in a covered country.

The author expresses the hope that the appended charts, or magnographs, which show how the character of the country rock and its changes, structural strike and dip and their changes, depth of burial and its variation, contact metamorphic effects, normal faults and thrusts, joint control of buried topography and other essential geologic features and phenomena, reflect themselves in magnetic observations. He believes that the charts will be of service in promoting magnetic methods and will inspire confidence in the probability that results obtained with a physical instrument may be interpreted with some degree of accuracy.--W. Ayvazoglou.

ZUR ANISOTROPIE DER PHYSIKALISCHEN PARAMETER VON GESTEINEN, SPEZIELL DER MAGNETISCHEN SUSZEPTIBILITÄT

(CONCERNING THE ANISOTROPY OF PHYSICAL PARAMETERS OF MINERALS, ESPECIALLY OF THE MAGNETIC SUSCEPTIBILITY)

By J. Koenigsberger

Zeitschrift fuer Geophysik, vol. 5, No. 2, 1929, pp. 62 - 64.

In the definition of the magnetic susceptibility, according to the methods suitable for field measurements, the crystalline slates have shown a distinct magnetic anisotropy similar to their thermic, electrical, and elastic anisotropy.

The anisotropy can also be explained without the acceptance of anisotropic bearers of magnetic induction, as for example in the case of pyrrhotite, by a certain arrangement of isotropic or disarranged anisotropic grains of minerals in rocks.--Author's abstract translated by W. Ayvazoglou.

A THEORY OF THE PERMANENT MAGNETIC FIELDS OF THE SUN AND EARTH

By Ross Gunn

The Physical Review, vol. 34, No. 2, July, 1929, pp. 335-344.

The motion of ions, executing short free paths under the influence of thermal agitation in an inhomogeneous magnetic field, in crossed magnetic and gravitational fields or in crossed magnetic and electric fields are shown to produce drift currents. The ion drifts are found to be opposite in direction to the drifts produced in analogous cases of long free path. Under the condition of radial symmetry and a closed circuit the magnetic gradient gives rise to circular currents which flow in such a direction that magnetic regeneration takes place. Regeneration is limited by demagnetizing currents arising from the thermal motions of the ions interacting with the resultant magnetic field and an internal electric or gravitational field.

The magnetic moments of the sun and earth are calculated from data which are approximately known and the correct magnitudes obtained.

The permanent fields arise from the thermal energy of the body and would be maintained if the bodies ceased their rotation. The asymmetry of the earth's magnetic field indicates that the hemisphere embraced by the Pacific Ocean is at a higher mean internal temperature than the rest of the earth.--Author's abstract.

INDUCTION-COEFFICIENTS FOR MAGNETOMETER-MAGNETS

By H. E. McComb

Terrestrial Magnetism, vol. 34, No. 3, Sept. 1929, pp. 241-247.

A special apparatus was designed for the investigation of induction-

coefficients of magnetometer-magnets by Lamont's method. Maximum accuracy is obtained when the vertical distance between magnets is equal to one-half the horizontal distance. Tests were made over wide ranges of horizontal and vertical distances.--Author's abstract.

PROCEEDINGS OF THE MAGNETIC SECTION OF THE INSTITUTE OF PRACTICAL GEOPHYSICS IN 1927

(In Russian)

By I. M. Bahurin

Bulletin of the Institute of Practical Geophysics, No. 4, 1928, pp. 181 - 196.

Magnetometric investigations were made in 1927 for the purpose of testing and comparing the deflection magnetometer constructed by the Institute of Practical Geophysics with the Tiberg-Thalen's magnetometer calibrated in the Institute of Practical Geophysics; the results obtained by observations with the deflection with the deflection magnetometer were also compared with those obtained by Schmidt's balance.

The work was carried on in the Zaporojsky and Pervomaisky districts. A map of the magnetometric survey and four profiles, showing the results of investigations in the form of the equipotential lines of the vertical component of anomalies is given. At some point of the investigated anomalies the vertical intensity Z reached a value equal to 1.340. About 1,000 observations were made. More exact results were obtained after the calibration of Tiberg-Thalen's apparatus in the laboratory of the Institute of Practical Geophysics.

Comparison of the apparatus previously mentioned showed a constant and more or less considerable difference between the data obtained by them. The reasons are not established yet.

In working with the deflection magnetometer constructed by the Institute of Practical Geophysics it was possible to define H and Z in about 40 to 50 points per day.--W. Ayvazoglou.

AN ESSAY OF COMPARISON OF THE RESULTS OF MAGNETOMETRIC AND GRAVIMETRIC SURVEYS OF DEPOSITS OF FERRUGINOUS QUARTZITES IN THE DISTRICT OF KRIVOY ROG

(In Russian)

By S. K. Ghirin and A. A. Popov

Bulletin of the Institute of Practical Geophysics, Leningrad, No. 4, 1928, pp. 299-308.

The authors describe the results of the magnetometric method of researches made in the district of Krivoy Rog over the same area previously investigated by the gravimetric method (see Lee, F. W., Geophysical Abstracts

No. 1, "Gravimetric researches of ferruginous quartzites in the region of Krivoy Rog." I.C.6120, Bureau of Mines, 1929, 25 pp.) in order to compare the data obtained by the two methods.

Tiberg-Thalen's and Thomson-Thalen's instruments were used. The instruments were calibrated at the magnetic station of the Institute of Practical Geophysics in Leningrad.

A map with curves showing variations in intensity of the vertical component across the strike of the deposits is given, and a geological map based on the data obtained by the gravimetric investigation made over the same area is added.

From the comparison of the two maps the authors draw the conclusion that the results of both the surveys are on the whole similar. In both cases the three main strata of ferruginous quartzites were determined.

In comparing the two methods the authors are of the opinion that in searching for iron-ores a preliminary estimation of the deposits should be made by the magnetometric survey, after which a more detailed gravimetric investigation should be carried out. Of course this conclusion concerns to the given case only.--W. Ayvazoglou.

3. SEISMIC METHODS

NEUE ERGEBNISSE DER EISSEISMIK

(NEW RESULTS OBTAINED IN THE SEISMICS OF ICE)

By H. Mothes

Zeitschrift fuer Geophysik, vol. 5, No. 3-4, 1929, pp. 120-145.

In July and August, 1928, artificial seismic waves were produced in Hintereisferner (Oetztaler Alps) by explosions and were measured optically by means of a vertical seismograph at distances of 180 to 600 meters. According to the mean figures of a great many single measurements the following values for the velocity of wave propagation have been found: in lingual ice, V_l = 3,600 m./sec.; in perpetual ice, V_l = 3,140 m./sec.; (V_l is the propagation of longitudinal waves running directly from the place of explosion to the place of observation). in lingual ice, V_{ll} = 1,690 m./sec.; in perpetual ice, V_{ll} = 1,350 m./sec.; (V_{ll} is the propagation of probably transverse waves following the same path).

In 25 cases reflections of longitudinal waves at the lower boundary surface of the ice, which served for the determination of the thickness of the ice, could be registered.

The reality of the reflections has been proved by the picture of the travel-time curves, as well as by the indication that the longitudinal reflection waves are subjected to total reflection. The results were confirmed and

extended during the ice-seismic work in March, 1929, on the Konkordia-Platz (Aletschgletscher).

A great number of pictures, as well as several tables showing the comparative data of the results of measurements, are added.--Author's abstract translated by W. Ayvazoglou.

SEISMISCHE BEOBACHTUNGEN VON STEINBRUCHSPRENGUNGEN

(SEISMIC OBSERVATIONS CONCERNING BLASTING IN QUARRIES)

By E. Wiechert

Zeitschrift fuer Geophysik, vol. 5, No. 3-4, 1929, pp. 159-162.

To determine the travel-time curves, a complete list of the literature available from 1925 to 1929 on seismic work concerning blasting in quarries has been collected by Wiechert. The purpose of the work consisted of studying the uppermost layer of the earth's crust. Wiechert started this work and after his death the work was completed under the direction of B. Brockamp. A table showing the data for these travel-time curves, called "Göttinger Laufzeitkurven," is given in the article. Two travel-time curves are drawn: P_2 for the longitudinal waves propagated through the primitive rocks (Grundgebirge"), and P_3 for waves having greater velocity and moving through the layers situated below the "Grundgebirge."

A brief description of the formation of the underground at the places of blasting and observation is added.--W. Ayvazoglou.

BEMERKUNGEN ZU DEN BEOBACHTUNGEN BEI STEINBRUCHSPRENGUNGEN

(REMARKS CONCERNING THE OBSERVATIONS MADE ON BLASTING IN QUARRIES)

By B. Brockamp and K. Wölcken

Zeitschrift fuer Geophysik, vol. 5, No. 3-4, 1929, pp. 163-171.

This is an article written in connection with the previous research work on blasting in quarries (see Wiechert's article preceding). After mentioning the apparatus used for observations, the authors examine the velocity of the longitudinal waves at different depths. The results of these observations are given in a table. The mean value for the velocity V_2 of the waves P_2 moving through the "Grundgebirge" is calculated to be: $V_2 = 5.90$ km./sec. Another table shows the time-curves for P_3, as well as differences of $P_2 - P_3$. The mean value of the velocity V_3 of the waves P_3 is: $V_3 = 6.72$ km./sec.

The determination of the depth h of several layers through which the waves are moving is given in the last part of the article. The depth h is calculated according to two formulas:

(1) $h = \frac{\triangle^s}{2}\sqrt{\frac{V_2-V_1}{V_2}}$ . . . . . Under the condition that the rays move upwards vertically.

(2) $h = \frac{\triangle^s}{2}\sqrt{\frac{V_2 - V_1}{V_2 + V_1}}$ . . . . . Under the condition of oblique rising of the rays.

Here $\triangle^s$ means the distance at which the time-curves intersect, and V_1, V_2 (or respectively V_2, V_3) the velocities in the corresponding layers. --W. Ayvazoglou.

ADAPTATION OF ELASTIC-WAVE EXPLORATION TO UNCONSOLIDATED STRUCTURES

By Frank Rieber

American Institute of Mining and Metallurgical Engineers, Geophysical Prospecting, 1929, pp. 654-667.

The article deals with the development of new types of apparatus and a number of extensions of the eariler mathematical methods of treating the results by which the elastic-wave methods may be used in a wide variety of locations and in relatively unconsolidated sedimentary materials to give a very accurate picture of structural conditions.

Examination is made of the development of the chief requirements of apparatus, such as the most sensitive means for recording vibrations and the most accurate methods for estimating the elapsed time between the explosion and the recording of the resulting vibration as it arrives at a distant receiving instrument. The equipment necessary for field operation is described.

In the last part of the article, Rieber explains the interpretation of the results, the construction of graphs, and the development of formulas. The article is illustrated by photographic pictures and diagrams.--W. Ayvazoglou.

RAYLEIGH-TYPE WAVES PROPAGATED ALONG AN INNER STRATUM OF A BODY

By K. Sezawa and G. Nishimura

Bulletin of the Earthquake Research Institute, Tokyo, vol. 5, 1928, pp. 85-91.

The method described in this paper resembles that used in the investigation of Rayleigh-waves on a stratified body. It is noted, however, that the authors have considered cases of both symmetric and asymmetric waves with respect to the central plane of the stratum. The important results are enumerated below:

1. Rayleigh-type waves propagated along an inner stratum of an elastic body can be classified into two elementary types. According to the symmetric or asymmetric forms of the waves about the central plane of the stratum, the waves may be designated as symmetric as asymmetric type.

2. Rayleigh-type waves propagated along an inner stratum show a generally dispersive nature.

3. The asymmetric type has a velocity higher than that of the symmetric type for the same values of L/H.

4. Velocities of wave propagation along a stratum are limited between those of distortional waves in the stratum and the outer medium.

5. A smaller ratio of $\frac{\mu'}{\mu}$ gives a longer range of L/H, which keeps the values of the velocity of propagation approximately constant.

6. As $\frac{\mu'}{\mu}$ tends to be very small, the constant value mentioned approaches the velocity of propagation of Rayleigh-waves on a semi-infinite solid body.

7. Energy of waves of short length accumulates in the vicinity of a weak stratum.---Author's abstract.

4. ELECTRICAL METHODS

SOME EARTH RESISTIVITY MEASUREMENTS

By F. W. Lee, J. W. Joyce, and Phil Boyer

Information Circular 6171, Bureau of Mines, Oct., 1929, 16 pp.

Owing to the increasing importance of resistivity measurements made on the surface as an index of underground structure and the difficulty of direct application of a simple mathematical system to the analysis of these measurements, the authors are of the opinion that, unless a new mathematical tool to meet these needs is invented the results must rest for the present upon extensive experimental work, rather than upon possible theoretical deductions. The purpose of this information circular is to set forth some difficulties encountered in solving this problem.

After the discussion of the system of determining the average resistivity of the earth and the factors influencing this resistivity, illustrated by a series of diagrams, the results of the equations derived are shown graphically in a figure.

The second part of the article deals with the "resistivity about a single electrode"; the method consists of using two current terminal connections, usually iron stakes, for connecting current to the ground. The results are presented in a table showing maximum variation of symmetry around a stake electrode.

Another table gives the computation of resistance at various radii between concentric hemispheres. Methods for measuring earth resistivity, surface and structure effects on resistivity, comparisons of measurements, and

differentiative resistivity measurements are explained and illustrated by many figures.

The authors conclude that by choosing the points of observation at close enough intervals it is possible to follow the contour of the various geological beds.

Of course considerable skill is required for the proper interpretation of the results.--W. Ayvazoglou.

TENTATIVE METHOD FOR MAKING RESISTIVITY MEASUREMENT OF DRILL CORES AND HAND SPECIMENS OF ROCKS AND ORES

By M. W. Pullen

Information Circular, 6141, Bureau of Mines, June, 1929, 11 pp.

As the determination of electrical resistivity of the earth may furnish information on the location of ore bodies of economic importance, the knowledge of this resistivity of various rocks is considered to be important to the geophysicist. The author presents a tentative method and gives the results of tests of drill cores from the Mineville magnetite district and of hand specimens of serpentine and chromite. Pullen describes (1) the method used for hand specimens, (2) resistivity measured with direct current, (3) resistivity measured with alternating current, (4) method used for diamond-drill cores, (5) resistivity measurement of drill cores at various depths, and (6) the effect of moisture on resistivity of drill cores.

Tables are given showing the effect of time on resistivity, the effect of direct current voltage on resistivity, the effect of alternating current voltage on resistivity, the resistivity measurement of chromite, and the resistivity at various depths. Eight figures are added.

The results of the tests are summed up by the author as follows:

"1. Surface leakages of current must be reduced for reliable resistivity measurements.

"2. Resistivity measurement is associated with electrical polarization, and the factor of time therefore enters as an element in its determination.

"3. The rate of polarization differs in various rocks.

"4. Variation of resistivity is a function of applied voltage and frequency.

"5. Because of low values of measured current, d.c. instruments must be used in a. c. circuits when measuring alternating current resistivity.

"6. There is a difference between alternating-current and direct-current resistivity.

"7. A definite method is proposed for preparing drill cores and hand specimens for resistivity measurements.

"8. Rock specimens of the same classification from different horizons of a drill core may show a wide variation of resistivity due to the latitude in the rock classification.

"9. Water content may greatly alter the resistivity of materials.

"10. Resistivity of a conducting ore body may be high, if the particles of ore are disseminated.

In conclusion the author invites discussion upon the electrical resistivity measurement methods in order to establish a uniform procedure and to adopt a system of rock-resistivity classification.--W. Ayvazoglou.

DIE GEOELEKTRISCHEN UNTERSUCHUNGSMETHODEN MIT WECHSELSTROM NACH DER SONDENMETHODE

(GEOELECTRICAL EXPLORATION METHODS WITH ALTERNATING CURRENT ACCORDING TO THE PROBE METHOD)

By P. Hülsenbeck and W. Geyger

Montanistische Rundschau, vol. 21, No. 1, August 1, 1929, pp. 289-295.

After mentioning Schlumberger's method for discovering useful deposits the authors draw attention to the advantage of using alternating instead of direct current.

A description of electrical methods of prospecting, accompanied by figures, is given. The difficulty of interpreting the curves obtained is explained and the necessity of studying the fundamental differences between the indications obtained on the surface of the ground and those at a depth are noted. The authors conclude that in order to secure good results a fundamental knowledge of the geology of the area under investigation is necessary.--W. Ayvazoglou.

METHOD FOR MEASURING THE SUSCEPTIBILITY OF ROCKS

By J. G. Koenigsberger

Terrestrial Magnetism, vol. 34, No. 3, 1929, pp. 209-214.

Lord Kelvin's mathematical method of electrical images applied to a magnet gives the susceptibility of a body having a plane surface placed near the magnet. The different corrections for varying distance, finite surface area, and finite height perpendicular to the surface are discussed. The method is suitable for use in the field with a variometer to determine the susceptibility of samples of rocks. Crystalline schists and other rocks often exhibit a magnetic anisotropy. A provisional scheme for a magnetic-geological classification of rocks based on remanent magnetism is given. The order of

magnitude of remanent induced KF (K, susceptibility; F, total intensity) is compared with present values.--Author's abstract.

EARTH-RESISTIVITY MEASUREMENTS IN THE LAKE SUPERIOR COPPER COUNTRY

By W. O. Hotchkiss, W. J. Rooney, and James Fisher

American Institute of Mining and Metallurgical Engineers, Geophysical Prospecting, 1929, pp. 51-65.

During the summer of 1927 the Department of Terrestrial Magnetism of the Carnegie Institution of Washington joined with the Michigan College of Mining and Technology in conducting a series of earth-resistivity measurements in the Michigan copper country. The geological structure in this section is quite varied and is fairly well known.

The purpose of the investigation was to discover the correlation between the variations in electrical resistivity of large masses of soil and rock in place and the geological structure below the surface. The work done may be roughly divided into two parts: (1) Measurement and study of the variations in resistivity in depth, or with volume, to establish such general relations as exist between the two; and (2) the determination of the specific resistivity of various formations in order to make available more fundamental data which may aid in the interpretation of the results of the former type of measurements.

A report by Mr. Rooney, which gives in detail the measurements made and the results obtained, forms the basis for this article.--Authors' abstract.

ELECTRICAL PROSPECTING FOR MOLYBDENITE AT QUESTA, NEW MEXICO

By Karl Sundberg and Allan Nordstrom

American Institute of Mining and Metallurgical Engineers, Geophysical Prospecting, 1929, pp. 125-137.

A survey is described which was carried out during October and November, 1927, by the Swedish American Prospecting Corporation at the Questa mine.

After briefly discussing the geological conditions at Questa and the exploitation of the molybdenite, the authors describe the "electromagnetic compensator method used for the survey." This method comprises the determination at every point of strike and dip of the polarization plane and the amplitute of real and imaginary parts in micro-gauss per ampere of horizontal and vertical components. The essential instrument used in such a survey is called the compensator; therefore the authors refer to this survey as a survey by the compensator method. The compensator used was invented by Karl Sundberg and E. D. Lindblom.

For reconnaissance work the authors used more simple methods, called the two-frame and three-frame methods, invented by Sundberg.

The results obtained by this survey have shown the possibility of using electromagnetic prospecting for finding small veins with poor conductivity in country of very rugged topography.--W. Ayvazoglou.

OPERATING PRINCIPLES OF INDUCTIVE GEOPHYSICAL PROCESSES

By J. J. Jakosky

American Institute of Mining and Metallurgical Engineers, Geophysical Prospecting, 1929, pp. 138-176.

All electrical geophysical methods depend for their operation upon the effects produced by the flow of an electric current. By studying these effects it is possible to predict the general axis of current flow. The greater flow of current is in the path of greatest effective conductivity; and since the effective conductivity of a mineralized zone is different from that of its surrounding envelope (usually much greater), it is possible to locate such a mineralized zone by the distribution of current. Due consideration must be given, of course, to geological structure, to type of mineralization, and to other factors. A number of methods are used to cause this flow of current, and various methods are available for detecting the presence of such current flow and for studying its effects. In a rather detailed way, this paper discusses the theoretical and practical phases of the inductive methods of geophysical prospecting.--Author's abstract.

FIELD OBSERVATIONS OF ELECTRICAL RESISTIVITY AND THEIR PRACTICAL APPLICATION

By J. G. Koenigsberger

American Institute of Mining and Metallurgical Engineers, Geophysical Prospecting, 1929, pp. 221-237.

The electrical specific resistance of rocks in the field is measured by sending a current through a medium of great volume, compared to the electrodes, whose resistivity should be measured. The whole resistance can be determined by the Wheatstone bridge and the specific resistance calculated by a well-known formula. The resistivity of some rocks in place in Germany and Switzerland is given--first, of a rock near the surface, and especially near electrodes.

Practical applications are made in the determination: (1) of the continuity of a conducting ore body, (2) of the surface area of an ore body, and (3) in the detection of a water-bearing fault in a salt mine.

Preliminary observations are described: (1) on the influence of the depth of ground-water table, and (2) on the detection of gas under high pressure in coal seams.

The observations in this paper of the electrical resistivity of rocks and soil in place are along the lines of the very interesting method and results of the Messrs. Gish, Rooney, Hotchkiss, and Fisher ("Earth-resistivity

measurements in the Lake Superior copper country," Amer. Inst. Min. and Metall. Eng., 1929, pp. 51-65.) All the work described has been done since 1920.--Author's abstract.

UEBER DIE TIEFENWIRKUNG BEI GEOELEKTRISCHEN RAHMENMETHODEN

(THE EFFECT OF THE GEOELECTRIC FRAME METHODS WITH REGARD TO THE DEPTH OF AN ORE BODY)

By J. N. Hummel

Zeitschrift fuer Geophysik, vol. 5, No. 2, 1929, pp. 72-80.

The magnetic field of a stationary current sent through the underground is examined for proof that rotation bodies of deviating conductivity are deposited in isotropic mediums which otherwise are homogeneous, whereby the axis of rotation is situated in the direction of the homogeneous field of current.

It is possible to show that in deposits situated at a great depth the disturbances produced in a normal regular magnetic field on the surface disappear almost at the same time with the distortions in the corresponding potential line figure.--Author's abstract, translated by W. Ayvazoglou.

DER SCHEINBARE SPEZIFISCHE WIDERSTAND

(THE APPARENT SPECIFIC RESISTANCE)

By J. N. Hummel

Zeitschrift fuer Geophysik, vol. 5, No. 3-4, 1929, pp. 89-104.

Some geophysical methods are based on the determination of the "apparent specific resistance."

In this paper the conception of the apparent specific resistance is explained, different ways for its disclosure are given, and the theory of the corresponding methods of solution is developed. These methods have proved to be suitable for the discovery of the disturbing bodies; but their real function is to indicate the character of the subsoil formed by horizontal layers. Calculations are given for two special cases and the results are discussed.--Author's abstract, translated by W. Ayvazoglou.

MODELLVERSUCHE MIT DER VIERTELWELLENMETHODE

(MODEL EXPERIMENTS WITH THE METHOD OF QUARTER-WAVES)

By J. N. Hummel

Zeitschrift fuer Geophysik, vol. 5, No. 3-4, 1929, pp. 104-118.

With the method of quarter-waves the proceedings take place in a zone close to or between a swinging dipol-antenna. Therefore all the three

members of Hertz's radiation equation must, in the theory, be taken into consideration for the dipol. The reaction of a stratum of very good conductivity upon the sender is calculated from this point of view.

In making model experiments, extremely short waves are produced according to the Barkhausen and Kurz methods. The reaction of a stratum of good conductivity upon the sending dipol is measured experimentally and the curve of the antenna current is drawn. If the simplified assumptions made during the calculations are taken into consideration at comparison, a good conformity between the theory and the practice is obtained.--Author's abstract, translated by W. Ayvazoglou.

CALCULATION OF COMPARISON OF HOMOGENEITY OF FIELDS OF SCHLUMBERGER, LUNDBERG, AND PETROWSKY

(In Russian)

By R. Skariatin

Bulletin of the Institute of Practical Geophysics,
No. 4, 1928, pp. 256-265.

Contents of the article:

1. Advantages concerning the different methods of placing the electrodes.
2. Calculation of Schlumberger's field.
3. Calculation of Lundberg's field.
4. Curves of homogeneity.

Owing to the fact that Schlumberger's field has, notwithstanding its great simplicity, one big disadvantage, that of heterogeneity of the electric field obtained, Lundberg's and Petrowsky's fields are used almost exclusively in the U. S. S. R.

As the calculation of Petrowsky's field has already been given in a previous article, "Calculations of an artificial electric field" (see Geophys. Abs. No. 1, I.C. 6120), Skariatin gives here only calculations concerning Schlumberger's and Lundberg's fields. The calculation of Schlumberger's field has been made according to Petrowsky's formulas and tables.

The calculation of Lundberg's field, which is somewhat more complicated, has been made by the application of the potential numbers given in a special appendix. The equipotential lines obtained are shown in a figure in which it can be seen that Lundberg's and Petrowsky's fields differ very little only.

In the last part of the article the question of the homogeneity of the three fields is discussed, and, based on the curves of homogeneities given

in the article, the author concludes that the most homogeneous field is that of Petrowsky. Lundberg's field differs very little; that of Schlumberger, of course, can not be compared with the other two because the curves of homogeneity have another form which is caused by the quite different method of setting the electrodes.--W. Ayvazoglou.

5. RADIOACTIVE METHODS

THE ABSORPTION OF PENETRATING RADIATION

By L. H. Gray

Proceedings of the Royal Society
Series A, vol. 122, No. A790, pp. 647-668.

Adopting the hypothesis that penetrating radiation is a type of γ radiation, its absorption in the atmosphere is investigated from the theoretical standpoint.

1. An attempt is made to establish a quantitative correlation between the ionization produced in an electroscope by penetrating radiation and the scattering absorption coefficient of the radiation.

2. The relative magnitudes of the ionization produced by primary and scattered radiations at any point are computed by approximate methods. The calculations are made on the basis of the Compton scattering formulas, and also on the basis of the new scattering formulas proposed by Klein and Nishina.

3. On the basis of these results the relation between the apparent absorption coefficient of a homogeneous isotropic radiation and the true scattering coefficient of the primary radiation is discussed.

4. The adequacy of the Klein-Nishina formulas to account for the observed absorption and scattering of γ - rays is briefly considered.--Author's abstract.

THE RELATION OF RADIO PROPAGATION TO DISTURBANCES IN TERRESTRIAL MAGNETISM

By I. J. Wymore

Proceeding of the Institute of Radio Engineers,
vol. 17, No. 7, July, 1929, pp. 1206-1214.

This paper presents the results of a study of an apparent interrelationship between radio reception and changes in the earth's magnetism. The results show that for long-wave daylight reception over great distances (4,000 to 7,100 km.) there is in general a variable but definite increase in

the intensity of the received signal following the height of severe magnetic disturbance. This increase reaches its maximum in from one to two days and disappears in from four to five days. For moderate distances (250 to 450 km.) there is an increase in the intensity of the received signal noticeable before as well as after the magnetic storm reaches a maximum.

These changes in intensity cover periods from two to four days both before and after the magnetic storm reaches its height.--Author's abstract.

BERICHT UEBER DEN JETZIGEN STAND DER UNTERSUCHUNGEN UEBER DIE DURCHDRINGENDE HÖHENSTRAHLUNG

(REPORT ON THE PRESENT CONDITION OF THE RESEARCHES CONCERNING PENETRATING RADIATION)

By K. Wölcken

Zeitschrift fuer Geophysik, vol. 5, No. 3-4, 1929, pp. 151-159.

In the article Wölcken summarizes the results of a series of recent works concerning penetrating radiation. He classifies the works into three main groups:

1. Nature of penetrating radiation,

2. Absorption,

3. Changes with regard to time. (Zeitliche Aenderungen.)

A list of literature and a brief resume of articles mentioned, showing the progress attained in the domain of researches in penetrating radiations, is given.--W. Ayvazoglou.

6. GEOTHERMAL METHODS

DETERMINATION OF GEOTHERMAL GRADIENTS

By K. C. Heald

The Oil and Gas Journal, Dec. 5, 1929, pp. 90, 91, 191, 192.

This paper is a report of progress made in determining the relationship between variations in earth temperatures and oil-field structures.

The research has been undertaken by the American Petroleum Institute in order to determine whether or not chemical reactions between petroleum and other substances really are the causes of abnormally high temperatures--a theory which has been popular with many European students.

Different types of structures have been considered, such as the gentle folds of Oklahoma, the faulted monoclines of the Balcones fault zone of Texas, the compressed and crumpled strata of the California fields, as well as the extreme deformation that occurs around the salt domes of Texas and Louisiana.

The work, which has been in progress since 1927, has been centralized in areas where results will be most significant to the petroleum industry.

OKLAHOMA. - Temperatures were measured in wells extending from about 18 miles east of the town of Okemah to Oklahoma City. A temperature of 100° F. has been observed in a hole less than 1,500 feet deep near Okemah and at a depth of 4,100 feet near Oklahoma. The author notes, although it may be pure coincidence, that the rocks slope in the same direction as the 100° temperature line, at a rate only slightly more abrupt. Future work is necessary to establish or disprove this relation.

Local Structure. - The work in Oklahoma offered an opportunity to learn whether local structures--anticlines and synclines--in gently folded rocks are reflected by the temperatures in those rocks or not. Heald is of the opinion that Van Orstrand's earlier work in Wyoming and particularly on the Salt Creek anticline has been substantiated by McCutchin's work in Oklahoma, leaving no doubt that in steeply tilted beds such a reflection can be detected in many cases.

Chemical Action. - The work in Oklahoma has not yet progressed far enough to justify any definite statements on the theory that temperatures higher than normal will be found above oil fields because of heat released through chemical reactions involving oil.

Water Circulation. - The regional picture seems to support the conception that waters deeply buried in a syncline would move up the dip and would result in making the temperatures in the "up-dip" areas somewhat higher at a given depth below the surface of the ground than are those in the syncline whence the waters came.

Effects of Unconformities. - No particular study of unconformities has been made in Oklahoma, but conditions there are suitable for determining whether or not unconformities can be detected by temperature measurements.

Effect of Strong Faulting. - Whether the faulting activity may be detected by temperature measurements can not be determined by the Oklahoma work, since conditions there are not particularly favorable.

TEXAS. Stratigraphic Conditions. - Measurements now available almost, if not quite, justify the conclusion that the age of the sediments has nothing to do with the temperature conditions.

Period of Cooling. - It would seem reasonable to believe that the temperature of the rocks that had been exposed to cooling would be less than that of rocks that had never been exposed. No decisive opinion is yet justifiable.

Problem of Unconformities. - This problem is mentioned by the author as being a complex one and requiring more study.

Relation of Earth Temperature to Structure. - A regional picture constructed to show about half the State indicates a belt of comparatively high temperature along the Balcones--Mexia-Luling fault in which the temperature increases with depth at the rate of 1° F. for about 43 feet. In the central region that temperature increases 1° F. for each 50 to 60 feet. Farther west and northwest the average rate of increase is less than 1° F. for about 100 feet. The few data available for the great basin of west Texas show that the basin can not be outlined merely by studying the temperature conditions in wells drilled in and near it. For example, in two wells about 25 miles apart, the temperature in one increased at the rate of 1° F. for every 84 feet, and in the other 1° F. for every 137 feet.

Big Lake Field. - The structure of Big Lake field is clearly reflected by the temperature measurements.

Salt Domes Studied. - Four domes (three of them in the Gulf Coast region) were observed. The work has shown that, in so far as this type of structure is concerned, the presence or absence of oil has no determinable effect. General outlines of salt domes can, of course, be located by measurements in very shallow drill holes.

Concentration of Heat. - In neither area has the work been done in sufficient detail to justify conclusions as to the probable conditions responsible for temperature variations.

Regional Metamorphism. - The idea that the amount of pressure to which a region has been subjected should be reflected in the temperature is not supported by the work in Texas.

CALIFORNIA. Geologic Conditions. - The rocks involved in the California measurements represent a very short period of geologic time. In comparing the results of the California work with those obtained in the other States, the author expresses his preliminary impression that time is not a factor having an important influence on temperature variations in the earth.

Plan of Work. - The Santa Fe Springs field was chosen for exploration, and more than 30 wells have been measured in that field. The data secured in California must be considered dependable, as some of the measurements have been repeated; they were also retested by C. E. Van Orstrand, who used different instruments from those employed by A. J. Carlson.

Results. - A report giving details of the work has been prepared by A. Carlson and is in process of publication by the American Petroleum Institute.

Relation of Temperature to Petroleum. - The work in California has so far contributed nothing to help answer the question as to whether or not petroleum itself is directly or indirectly responsible for abnormal earth temperatures.

Relation of Temperature to Circulating Waters. - Although some of the fields of the San Joaquin Valley have been studied in detail so that it is possible to outline with reasonable accuracy areas in which certain sands contain waters that are moving downward from the outcrop and other areas where the waters are either standing still or are moving up towardsthe outcrop, no temperature work has been done in these areas as yet.--W. Ayvazoglou.

7

7. UNCLASSIFIED METHODS

GEOPHYSICAL INVESTIGATIONS AT CARIBOU, COLORADO

By C. A. Heiland, Charles W. Henderson, and J. A. Malkovsky

Technical Paper 439, Bureau of Mines, 1929, 45 pp.

This paper presents a series of articles on geophysical investigations written with the special object of determining which geophysical method or methods would be the most economical and successful in a particular district.

The United States Bureau of Mines in deciding to begin this study, thought it most advisable to select for this work a deposit that in part, or as a whole, was so exposed at the surface as to enable a thorough study to be made of the relationship of geologic factors to the physical effects produced by them.

The following articles are included in this technical paper:

1. Geology of the northern magnetite deposit, Caribou, Boulder County, Colorado, by C. W. Henderson.

2. Geodetic work at Caribou, Colorado, by C. A. Heiland and J. A. Malkowsky. (Plane survey: determination of astronomic north direction, arrangement of stations, topographic survey).

3. Relationship of topography, geology, and magnetic disturbance, by C. A. Heiland. (Method of topographic analysis; application of method at Caribou; agreement between topographic anomalies and earth-magnetic disturbances).

4. Earth-magnetic measurements, by C. A. Heiland and J. A. Malkowsky. (Instruments and procedure: Magnetic field balances, range of instruments, scale-value determinations, use of auxiliary magnets, determination of declination, corrections, computation of magnetic field, graphical representation. Conclusions drawn from magnetic maps: effect of magnetite distribution, factors affecting magnetization of deposits, extent and dip of deposit, discussion of causes of negative anomalies.)

Thirteen figures illustrate the articles, showing: (1) Geologic map of northern magnetite deposit, Caribou, Colorado; (2) topographic map and plan

of magnetometer and torsion-balance stations on the northern magnetite deposit; (3) derivation of topography from normal shape produced by erosion of homogeneous mass; (4) topographic gradients; (5) measurement of magnetic vertical intensity; (6) lines of equal anomaly in magnetic declination D; (7) lines of equal anomaly in magnetic vertical intensity $\triangle Z$; (8) lines of equal anomaly in magnetic horizontal intensity $\triangle H$; (9) total magnetic disturbance vectors $\sqrt{\triangle X^2+\triangle Y^2+\triangle Z^2}$ in isometric projection; (10) horizontal magnetic disturbance vectors $\sqrt{\triangle X^2+\triangle Y^2}$; (11) magnetic profile parallel to strike of deposit; (12) magnetic profile perpendicular to strike of deposit; and (13) magnetic profile perpendicular to strike of deposit.--W. Ayvazoglou.

THE FIGURE OF THE EARTH DERIVED BY TRIANGULATION-METHODS

By W. Bowie

American Journal of Science, vol. 18, No. 103, July, 1929, pp. 53-59.

Problems of determining the figure of the earth, as well as the results obtained by geodetical methods of measurement, are briefly discussed. Based on the fact that insufficiency of land area covered by triangulation, (less than 2 per cent of the area of the whole globe), as well as inadequate cooperation between the continents, constitutes the main weakness in the determination of the figure of the earth by means of triangulation and astronomic observations, the author expresses the opinion that better results may be reached after determinations have been made in each of the continents and also when the triangulation of all the continents can be combined in the determination of the figure of the earth.

The cooperation of the United States, Canada, and Mexico in geodetic operations is noted especially. Good results may be expected from this cooperation, as not only the triangulation of the three countries is connected but the same spheroid is used in the computations, that of Clarke of 1866 as expressed in meters.--W. Ayvazoglou.

SUR LA RECHERCHE GÉOPHYSIQUE DUE PÉTROLE EN ITALIE

(CONCERNING GEOPHYSICAL PROSPECTION FOR OIL IN ITALY)

By Arnaldo Belluigi

La Revue Pétrolifière, No. 346, No. 16, 1929, pp. 1617-1618.

Belluigi describes the work on oil prospecting done in the plain of the river Po. Four thousand torsion balance measurements and several thousand electrical measurements were made. Seven anticlines were established. Isostatic reduction has been carried out by pendulum measurements.

The author examines briefly some geophysical questions concerning prospecting for oil in the plain of the Po, such as: (1) The characteristics of the deep cores which showed gravimetric anomalies; (2) the problem of the form; (3) the problem of the perimeter; (4) the problem of the tectonic axis; (5) the question of the magnetic relief; (6) the question of the electric relief.--W. Ayvazoglou.

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MINING METHOD AND COSTS AT THE UTAH COPPER CO., BINGHAM CANYON, UTAH.

BY

A. SODERBERG

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February, 1930.

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHOD AND COSTS AT THE UTAH COPPER CO., BINGHAM CANYON, UTAH[1]

By A. Soderberg[2]

INTRODUCTION

This paper describing the mining practice at the Utah Copper Co., Bingham Canyon, Utah, is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts of the United States.

The Utah Copper Co. operates the low-grade open-cut copper mine at Bingham Canyon, Utah, which is 30 miles in a southwesterly direction from Salt Lake City. In addition to the mine the company operates two flotation concentrators, the Magma plant and the Arthur plant, which have a combined capacity at maximum metallurgical efficiency of 60,000 tons daily. The concentrators are at Magna, about 18 miles from the mine. The company operates its own standard-gage railway, known as the Bingham and Garfield Railway, between the mine and the mills (fig. 1). Concentrates are smelted at the Garfield smelter of the American Smelting and Refining Co., situated 4 miles from the mills.

ACKNOWLEDGMENTS

Acknowledgment for material and assistance received in the preparation of this paper is made to E. E. Barker, engineer of mines, H. C. Goodrich, chief engineer, Louis Buchman, assistant superintendent of mines, J. A. Marsh, mine geologist, and H. L. Garrity, safety engineer, of the Utah Copper Co.

HISTORY

The early history of the Utah Copper Co. has been published so many times that a brief review of its early beginnings will suffice for the purpose of this discussion. Long before the organization of the Utah Copper Co. it was known that large tonnages of low-grade sulphide ore existed at Bingham Canyon. As early as 1887 Colonel Enos A. Wall realized that the property had a potential value. In 1895 Captain Joseph R. DeLamar heard about the deposit and had it examined. A mill test was made and it was determined that a 60 per cent

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U S. Bureau of Mines Information Circular 6234."

2 - One of the consulting engineers, U. S. Bureau of Mines.

recovery could be effected. In 1898 Robert C. Gemmell examined the property and made a very encouraging report to his principals. Efforts to promote the property, failed, however, and it was not until 1903 that D. C. Jackling was able to obtain the necessary capital to purchase options from Colonel Wall. Among Mr. Jackling's backers were Charles M. MacNeil, Spencer Penrose, and R. A. F. Penrose. At this time the property was examined by F. H. Minard, who made another favorable report, and shortly afterwards the Copperton mill of 300 tons daily capacity was designed and constructed.

The Utah Copper Co. was first organized on June 4, 1903, under the laws of Colorado, with a capitalization of $500,000 and shares at $1 each; D. C. Jackling was given the position of general manager. Later, in April, 1904, the company was reorganized under the laws of New Jersey, and the capitalization was made $4,500,000 with shares having a par value of $10. In 1904 the Copperton mill began its operations with G. G. Janney, mill superintendent, in charge. It was on the basis of results obtained in this plant that the Magna mill was later designed.

At first, arrangements were made with the Denver and Rio Grande Railroad Co. to transport the ore from the mines to the mills, but in 1911 it was found necessary for the company to lay its own rails in order to assure a sufficient tonnage at all times for the mills.

The company acquired the Arthur mill in 1910 when the adjoining Boston Consolidated Co. mine was purchased. Since that time additions and improvements have taken place until the present daily capacity of 60,000 tons has been established.

The first shovel was placed in operation in 1906, with J. D. Shilling, sr., as mine superintendent.

GEOLOGY

The geology of the Bingham deposit is simple. Pennsylvania quartzite beds thousands of feet thick are intercalated with a few narrow limestone beds, and intruded by a mass of monzonite porphyry (fig. 2). The genesis of the commercial ore deposit is a typical example of secondary enrichment. The mineralization within the porphyry is the result of a period of intense action following the solidification of the mass; hot hydrothermal solutions caused the comparatively uniform dissemination of chalcopyrite, bornite, and pyrite. Subsequent secondary enrichment has increased the values; meteoric waters dissolved the copper in the upper portions, leaving a red capping, and redeposited it as chalcocite, covellite, and bornite. This mass of porphyry ore, when present reserves are added to the tonnage already mined, measures approximately 800,000,000 tons. The grade of the present reserves averages 1.066 per cent copper.

PHYSICAL CHARACTERISTICS

The Utah Copper ore body has its long axis in a northeasterly and southwesterly direction with an over-all length of about 6,000 feet, a maximum width of 4,000 feet, and a vertical depth of about 2,000 feet. Approximately an average of 115 feet of capping or completely leached porphyry covered the ore; in places, however, the sulphide zone was not

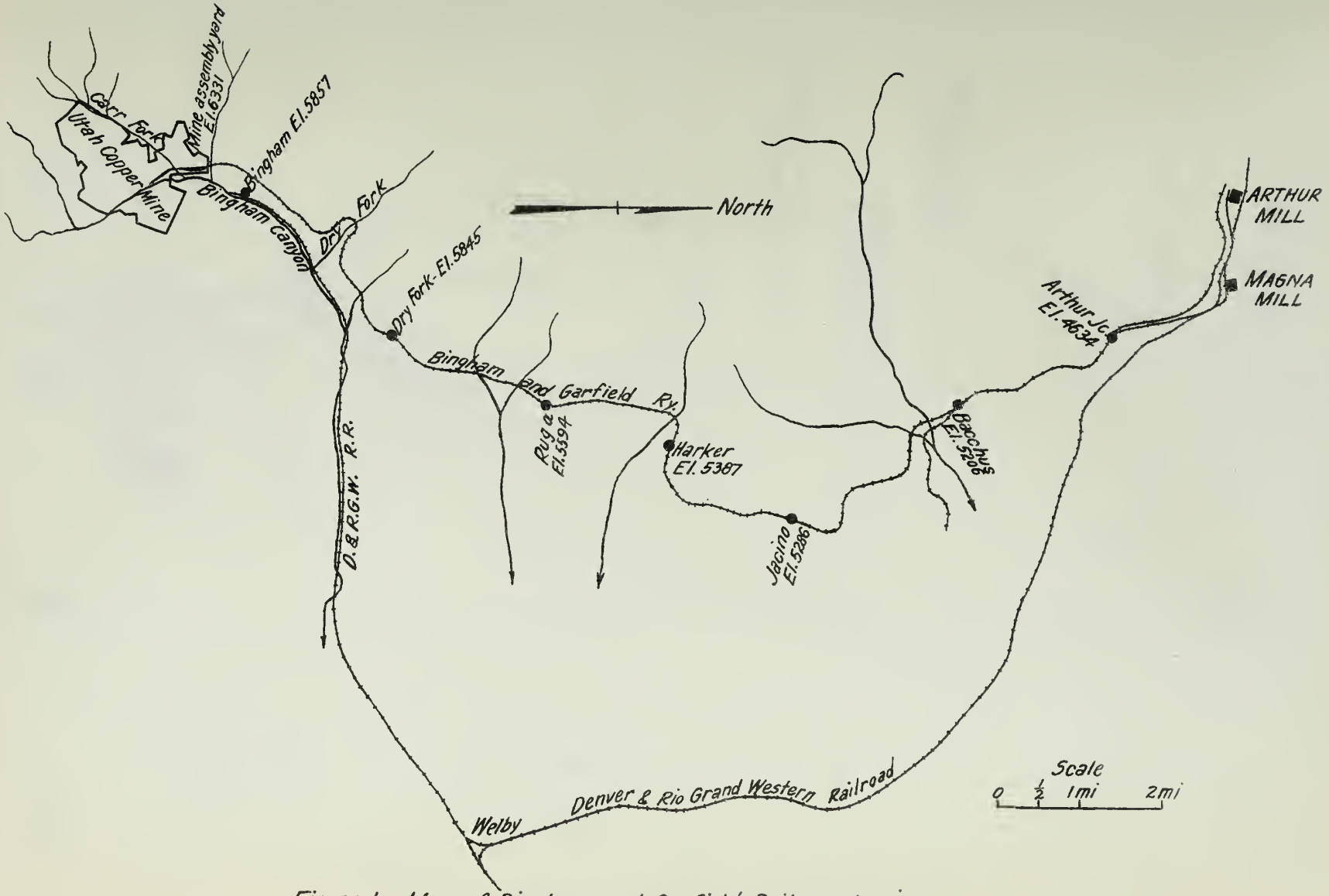


Figure 1. - Map of Bingham and Garfield Railway showing Utah Copper Company Mine and Mills

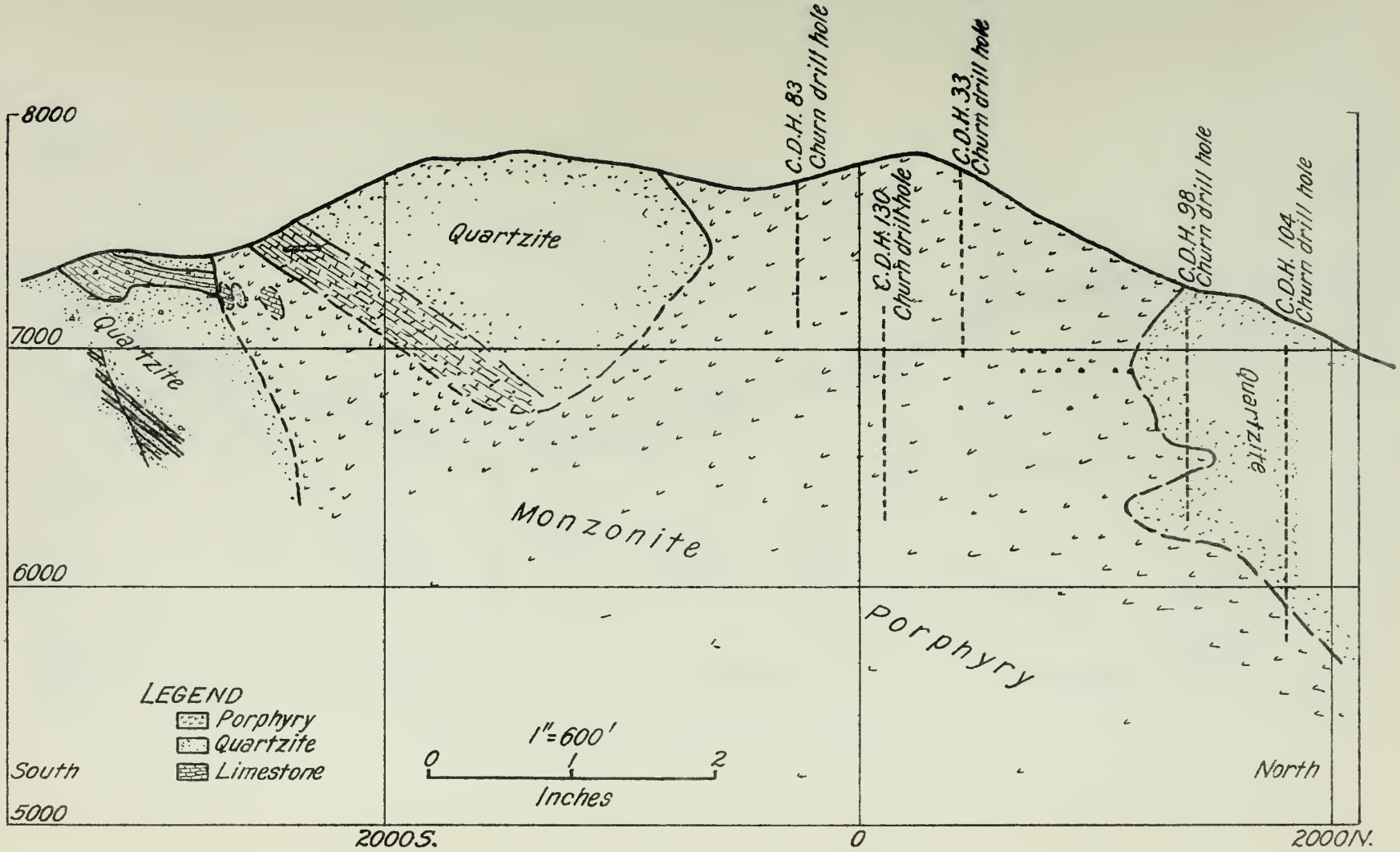


Figure 2.- Typical Geological Section

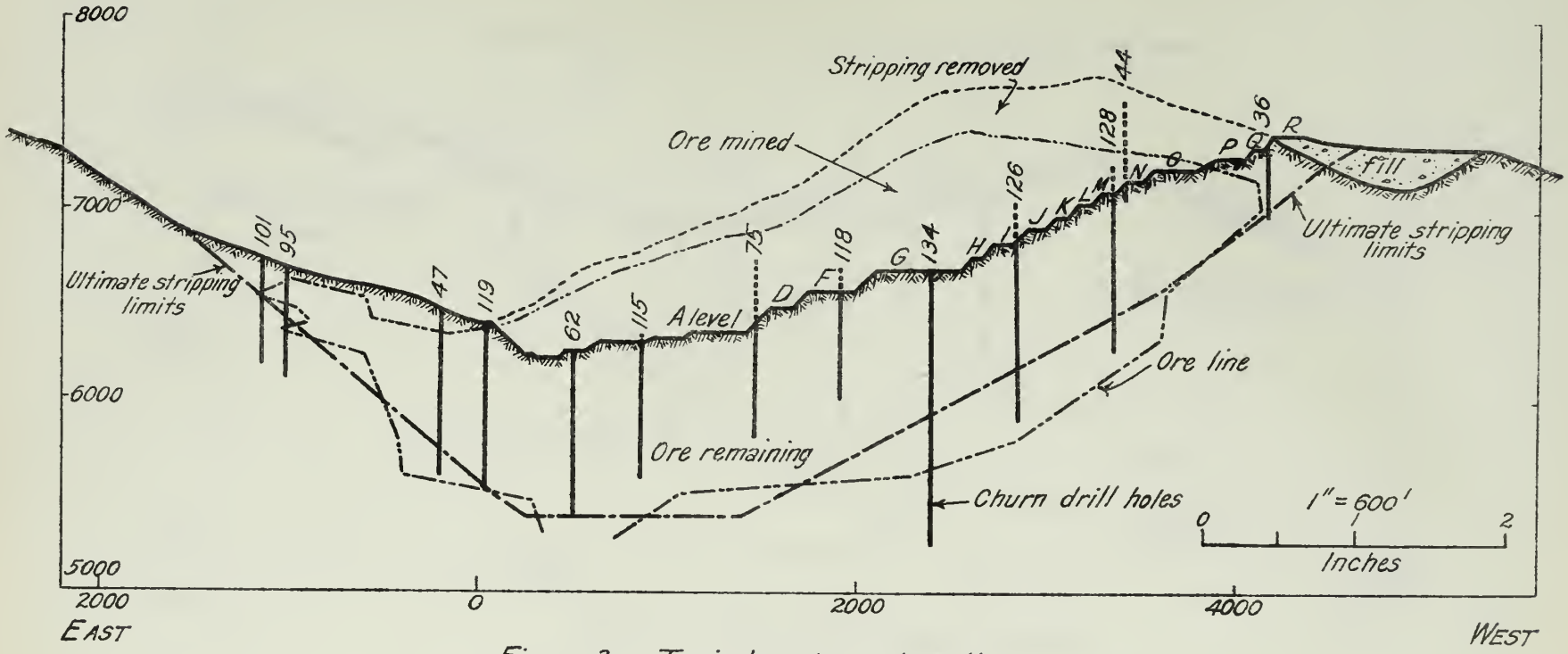


Figure 3. - Typical east-west section

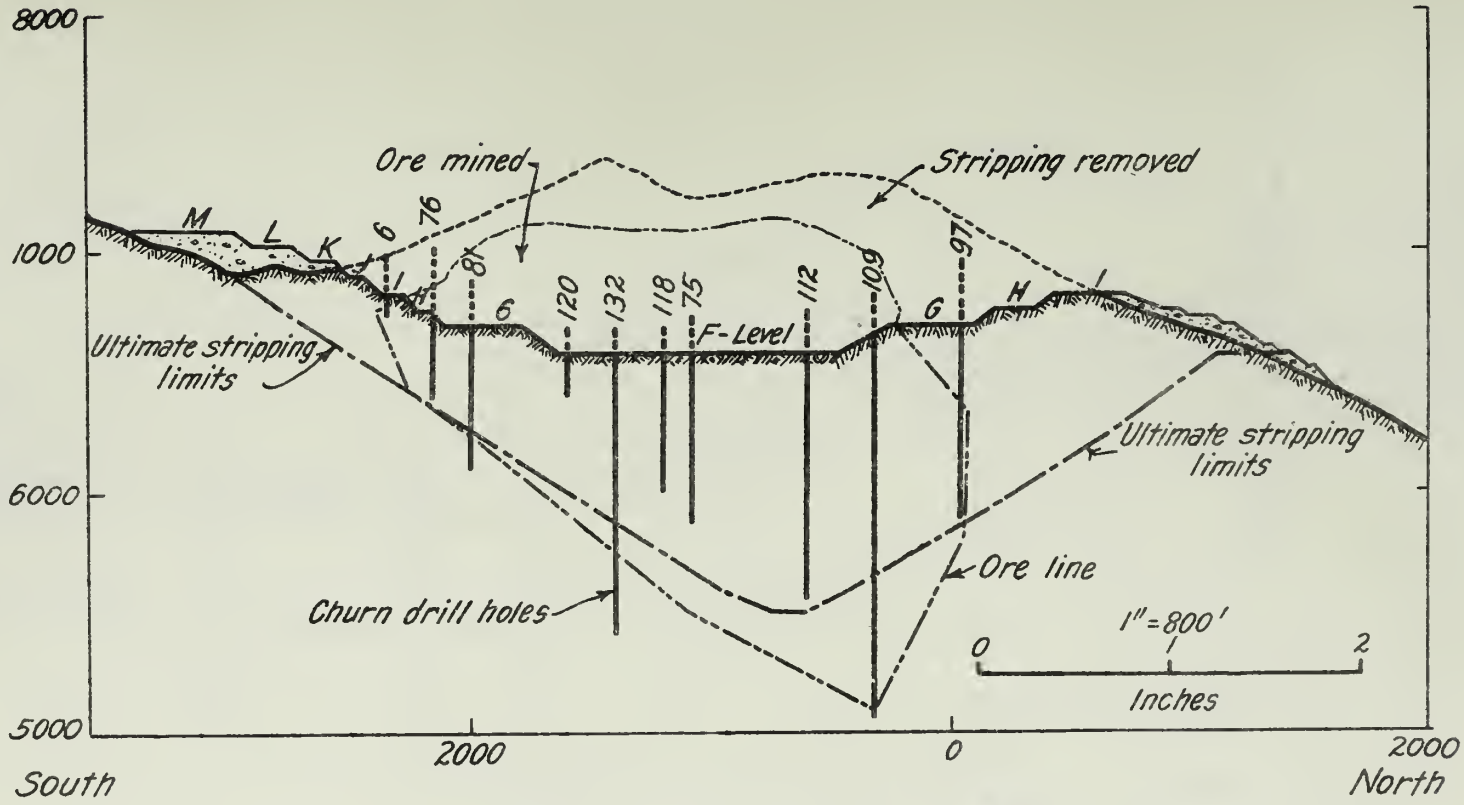


Figure 4. - Typical North-South Section

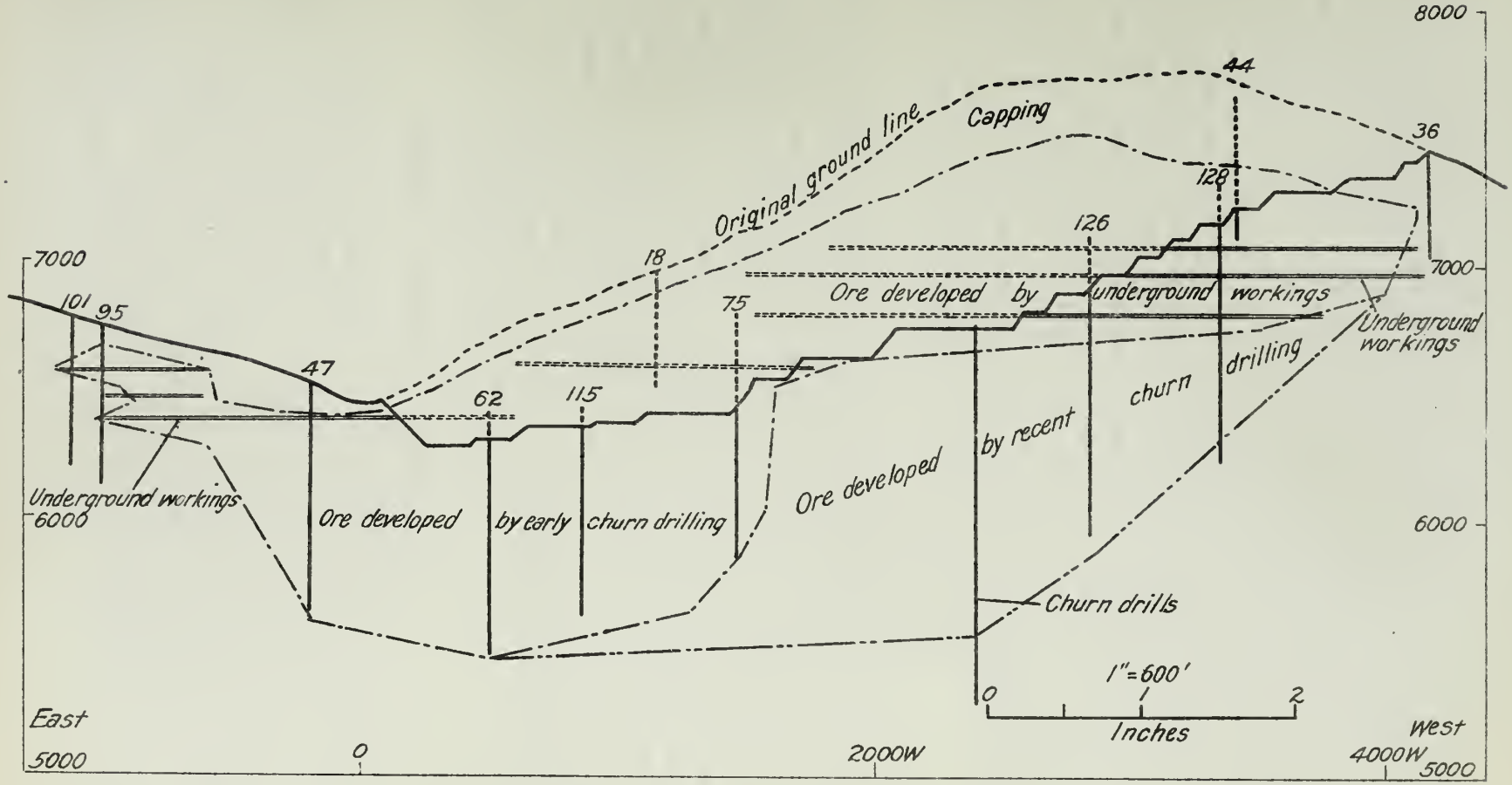


Figure 5-. Illustrative section showing comparative ore development by underground workings and churn drills

more than 20 to 50 feet from the surface. About 100,000,000 cubic yards of this capping has been removed. The ore is relatively soft and breaks easily along fracture planes into sizes that can readily be loaded by power shovels with a minimum of secondary blasting. Copper occurs in the ore as chalcopyrite (80 per cent), chalcocite (9 per cent), covellite (7 per cent), and bornite (4 per cent) uniformly disseminated throughout the intrusive mass.

It was found early that the ore body was one of magnitude and that the ratio of stripping to ore would not be more than one of stripping to two of ore; with these facts in mind it was decided from the outset to strip the overburden with steam shovels and then load the ore by the same method into cars for transportation to the mills. The angle of repose of the finished slope of a steam shovel mine is of prime importance, and it was soon indicated by churn-drill holes that the slope to the south (fig. 3) would favor shovel operations, in that the line of contact between commercial ore and waste was about 45° from the horizontal. To the north (fig. 4), however, the line of contact is much steeper, approximately 70° from the horizontal, and here the ore is overlaid by the quartzite beds. To mine the deep ore to the north, it was apparent that much waste which extended to depths of several hundred feet in this section of the mine would have to be removed. Although the stripping ratios are excessive in sections of the deposit, the average ratio at present does not exceed the removal of 1 ton of waste for every 2 tons of ore.

The grade of the ore, generally speaking, is very even, and while the commercial values range from six-tenths of 1 per cent to over 2 per cent there is little variation in the run-of-mine average from month to month, owing to the distribution of many digging units throughout the operating area. Within the confines of the area stripped, three varieties of mineralized porphyry are exposed. One is a dark basic porphyry confined to the south end of the levels. To the north of this basic porphyry is a series of prominent faults striking to the southwest and dipping to the north. In the proximity of the fault zone, the porphyry is highly silicified and has a greater mineralization than that found in the basic porphyry to the south. North of the fault zone or silicified porphyry is a typical gray monzonite porphyry which is the most highly mineralized zone of the mine and is also largest in extent. Although large tonnages of 2/10 to 3/10 of 1 per cent material occur, the quantity of so-called "possible ore" containing between 3/10 and 6/10 of 1 per cent copper is relatively insignificant, in other words, the material excavated for the most part is definitely either ore or waste, which is fortunate in a mining sense for the operator; Of course there are exceptions to this rule, but in general this is the case.

METHODS OF PROSPECTING AND EXPLORATION

Prior to the advent of steam shovels at the Bingham property, many miles of underground work had been driven by the Boston Consolidated and other companies operating in the district, so that in the beginning most of the proved tonnage was developed by this means. After the organization of the Utah Copper Co., both diamond drills and churn drills were used in exploration. Unsatisfactory results were noted from the very few diamond-drill holes drilled in the porphyry, as there was such a wide variation between core and sludge samples that diamond-drill work was discontinued and prospecting continued solely by means of the churn drill. This prospecting has, with the exception of short intervals, been continued to the present day. Figure 5 shows the relative development of the ore body by underground workings and churn-drill holes.

Two types of churn drills are used at present: One, a modified standard rig for drilling holes to depths of 1,000 feet or less, and two standard oil rigs for drilling the deeper holes; all are operated by electric power. When any considerable depth is anticipated, the holes are started with a 26-inch bit and a 26-inch stovepipe casing is carried down 80 or 100 feet, depending upon the extent to which the ground caves. At this point a 23-inch casing is inserted and carried down until it is "frozen" in the hole by caving ground. Underreaming is often resorted to when the casing will not follow the bit. Smaller sizes of casing are inserted as required, and in some instances casings as small as 4 inches have been used. It became a rule early in this work not to run an open hole more than 50 feet in advance of the casing, to eliminate the danger of salting the bottom samples by cavings from the bore of the hole above, and to minimize the need of underreaming, consequent delays, and increases in drilling cost. It is thoroughly understood that an accurate sample is wanted, even at the sacrifice of depth of hole if necessary.

All churn drilling is done by contract on a footage plus labor and supply cost basis. Following is a cost statement of one of the recent holes completed:

Cost of drilling hole with Standard electric rig

Spudded in with 23-inch casing.
Elapsed time, 455 days.
Average progress per day, 3.23 feet.
Average drilling progress per day, 3.92 feet.
Total footage drilled, 1,468.8.

	Cost	Per foot
Moving and setting-up drill:		
Labor, power, and water supply	$1,667.08	$1.14
Actual drilling:		
Amount paid contractor plus labor and supplies	19,578.33	13.33
Casing hole:		
Labor and supplies	4,468.47	3.04
Sampling and assaying	4,367.55	2.98
Total	$30,081.43	$20.49

METHODS OF SAMPLING AND ESTIMATES OF TONNAGE

Sampling at the churn drills is under the supervision of the geological department. A sampler is assigned to each rig.

Samples are taken every 5 feet and the cuttings removed by a suction bailer. The

CHURN DRILL HOLE NO. 134

Cordinates S. 973.75 W. 2396.39 El. 6708.0

Casing Left in Hole	Casing Record	Adopted Assays CU	Character of Material	General Remarks & Minerals
23" Casing 70' with shoe 20" " 100' " " 15½" " 323' " " 12½" " 130' " " 200	23" casing 20" casing	Elev. 6698	Silic. Porph.	8-14-28, Spudded in 23" Bit Pyrite, chalcopyrite, chalcocite, covellite, bornite, molybdenite
			Silic. Porph	Chalcopyrite, pyrite, chalcocite, covellite bornite, molybdenite
		0.88%	Silic. Porph	Chalcopyrite, pyrite, bornite, chalcocite
400	15½" casing 8¼" casing	Elev. 6393	Silic. Porph.	covellite, molybdenite Chalcopyrite, pyrite, chalcocite, bornite, covellite, molybdenite
600		1.16%	Silic. Porph.	Chalcopyrite, pyrite, chalcocite, molybdenite, bornite, covellite
800			Gray Porph.	Chalcopyrite, chalcocite, pyrite, covellite, bornite, molybdenite
1000	12½" casing	Elev. 5833	Dark Porph.	Chalcopyrite, chalcocite, pyrite, covellite bornite, molybdenite.
1200	10" casing	0.81% Elev. 5573 0.52% Elev. 5543	Limestone Dark Porph.	Altered, chalcocite, chalcopyrite, pyrite, covellite, bornite, molybdenite. Chalcopyrite, chalcocite, pyrite, covellite, bornite, molybdenite
1400	6¼" casing	0.78% Elev. 5468 0.54% Elev. 5423 0.80%	Limestone Mixed zone of limestone and porphyry Dark Porph.	Black, large amounts of chalcocite in limestone in porphyry primary minerals only, chiefly chalcopyrite, bornite, pyrite, unusal amount at bornite In limestone; chalcopyrite, chalcocite, covellite
		Elev. 5233	With limestone inclusions	In porphyry; chalcopyrite, bornite, pyrite, molybdenite 2-10-29 - Hole completed - Depth 1475'

Figure 6. – Condensed Churn Drill Log

content is discharged into a launder, passed over a screen 8 to 10 feet long, from which the large sizes are sent through a small gyratory crusher, then combined with the undersize, and the whole is passed into an inverted cone-shaped tank. Sludge in the tank is agitated for 20 minutes by a mechanical agitator that revolves near the bottom, aided by compressed air that enters through a 1-inch line near the bottom of the tank. From here the sludge passes to a cutter so constructed that three separate samples are obtained. One is sent to the mine assay office, one to the assay office at the mills, and the third and smallest sample is cut down to 5 pounds and placed in a 5-gallon wet sample can. This last sample is used as a part of a composite sample for every 100 feet drilled to serve as a check on the 5-foot samples; it is also used for making experimental flotation tests. Samples taken for immediate analysis are thoroughly dried on a large sheet-iron stove, care being taken not to burn or break down the sulphide. A 2-pound specimen is saved of every 25 feet of hole drilled in porphyry and of every 5 feet when the hole is near a porphyry quartzite contact. These specimens are examined by the geologist to determine the minerals contained and character of the rock.

Daily reports are made and a complete log of the hole is kept by the geological department (fig. 6). A record of the assay returns from the mine and mills is kept, and when a variation of over 0.05 per cent exists between the mine and mill assays, duplicate analyses are made of the pulp and averaged for the adopted assay.

Owing to the irregularity of the surface, holes were not drilled at the actual intersections of predetermined squares, but are drilled as nearly as possible at the corners of equilateral triangles. Where values are consistent, a spacing of 400 feet is considered safe, but as the limits of the ore body are approached, holes 200 feet apart have frequently been drilled.

In making the ore tonnage estimate, each level was treated as a separate mine; a plan map of each level was made showing the limits of the developed ore, underground workings, churn-drill holes perforating that particular level, and the ultimate location of the level. The specific gravity of the ore was determined, which gave a factor of 13 cubic feet per ton in place. The ore area was divided into blocks 100 feet square, with the height of the shovel bank in question taken as the depth of the block. Drill-hole assays within the segment of the hole between the top and bottom elevations of the bench were averaged, and the value was assigned to the 100-foot block perforated by the hole. The intervening blocks between drill holes were assigned assay values determined by proportioning the average of the intercepted drill-hole segments in accordance with their distances from the block in question. Where blocks were cut by the drifts and crosscuts of the old underground mine workings, the assays taken were also averaged for each block.

A detailed system of toe sampling is used at the mine to enable the operator to have a close check on the grade of ore loaded by each shovel. Samples are taken every 10 feet along the face of the level, following each shovel cut. The assay results are placed on a plan map showing the toe and edge of each level. New toe sample maps are made up every two weeks.

In the tonnage and grade calculations the toe assays were also averaged for each 100-foot block and compared with the assigned churn drill and underground mine assays in the same block. From this comparison a discount factor was arrived at for those levels where any decided difference was noted between toe assays and drill and underground mine assays, the toe assay being used as the basis for the discount. Differences in value were found to be confined largely to old stope areas, while the drill assays checked closely with the toe assays.

For estimating tonnage and grade of the ore located below the lowest level of the present working faces, namely 6,240 feet elevation, a plan map was made as of this elevation, showing all drill holes extending below this plane. Volumes were figured by triangular prisms bounded by drill holes, multiplying the area by the average depth of the hole and weighing the assays in accordance with the footage. Refinements of this method, using corrections for triangles that were not equilateral, were tried, but the differences did not warrant this procedure. The results obtained from the separate levels were combined with this lower calculation to give the total gross figure.

Preparatory to making detailed ore estimates, the cut-off between commercial ore and waste must be determined. In other words, a grade must be determined below which the material can not be mined and meet its mining and treatment costs and show a profit. To arrive at this cut-off grade, certain assumptions must of necessity be made, such as the selling price of copper, the estimated recovery in per cent of gross metal content, and the cost of producing a pound of copper, which includes all costs other than stripping. The stripping cost is kept separate for reasons that will develop later. Table 1 is set up to illustrate the method used to determine the point of cut-off:

Table 1.- Method of determining point of cut-off between ore and waste

(1) Per-cent copper	(2) Gross pounds per ton	(3) Assumed recovery, per cent	(4) Pounds recov-ered	(5) Assumed selling price per pound, cents	[1](6) Assumed cost per pound, less stripping, cents	(7) Partial net profit per pound, cents	(8) Loss per pound, cents
1.10	22	91	20.0	13.5	6.25	7.25	-
1.00	20	90	18.0	13.5	6.94	6.56	-
.90	18	89	16.0	13.5	7.81	5.69	-
.80	16	87	13.9	13.5	8.99	4.51	-
.70	14	85	11.9	13.5	10.50	3.00	-
.60	12	83	10.0	13.5	12.50	1.00	-
.50	10	80	8.0	13.5	15.62	-	2.12

Table 1.- Method of determining point of cut-off between ore and waste (Continued

(1) Per-cent Copper	(9) Stripp-ing cost per cubic yard	(10) [2]Ratio of waste to ore	(11) Proportionate stripping cost per pound this basis, cents	(12) (6+11) Total cost per pound, cents	(13) Profit per pound, cents	(14) Loss per pound,
1.10	$0.40	1/2 to 1	.50	6.75	6.75	-
1.00	.40	1/2 to 1	.56	7.50	6.00	-
.90	.40	1/2 to 1	.62	8.43	5.07	-
.80	.40	1/2 to 1	.72	9.71	3.79	-
.70	.40	1/2 to 1	.84	11.34	2.16	-
.60	.40	1/2 to 1	1.00	13.50	-	-
.50	.40	1/2 to 1	1.25	16.87	-	3.37

1 - Based on a cost of $1.25 per ton These figures can be varied to cover increasing costs due to increasing copper content, such as bullion freight, refining, selling, etc.

2 - Or one-fourth yd waste to 1 ton ore.

Under this set of conditions a copper content of .6 per cent would be the point of cut-off; any grade under this figure would be waste, and anything over should be classed as ore.

Other tables should be made with a variation of doubtful assumptions to assist the engineer in establishing a safe cut-off figure.

It is at this stage convenient to set up a table of grades showing the amount of stripping any given grade of ore will carry. This is usually worked up as follows:

If it costs 40 cents to waste a cubic yard of overburden weighing 2 tons, 1/2 ton will cost 10 cents. It then becomes necessary to determine what grade of ore will yield a return of 10 cents per ton under the given conditions, assuming the average recovery to be 85 per cent. The figure in this case is .0436 per cent copper, arrived at as follows:

.0436 per cent copper x 2,000 pounds = .872 pounds x 85 per cent recovery = .741 pounds;

.741 pounds x 13.5 cents equals 10 cents, the cost of moving 1/2 ton of stripping.

By adding this increment of grade (.044 per cent) to the ore, it will support the removal of an additional half ton of waste for each addition of the increment. One can then set up the following table:

Table 2.- Tonnage of stripping, plus all other costs, carried by 1 ton of ore of various grades

Grade of ore, per cent	Tonnage of stripping
0.6	0.5
.644	1
.688	1.5
.732	2
.776	2.5
.820	3

From the foregoing tables, graphs can be made from which it can be determined at a glance whether or not a certain block of material is ore or waste when the stripping ratio has been determined.

Dividends, of course, can not be paid on a grade of ore at or near the point of cut-off; the average grade, therefore, must be well in excess of the cut-off grade, and no section of the ore body that will not pay its own way should be combined with higher grades for the purpose of increasing the reserve. Possible exceptions to this rule appear, of course, when a "horse" of waste or a small amount of low-grade material occurs that has to be removed in any case. These small quantities of waste may not be easily separated and may be milled at a loss (capacity permitting) which, however, will be smaller than the cost of removing the material as waste. In such cases, the low-grade tonnage is included in the ore reserve with its grade. The engineer's judgment will guide him (after he has made a complete analysis of the ore body) in rounding out an estimate where so many variables are concerned. It is well to remember that material which at the time of the estimate is waste, may come into the classification of ore by an increase in the price of copper, by an improvement in metallurgy, or by a lowering of costs with improved equipment.

CHOICE OF METHOD

A point that is never lost sight of is that the total cost of mining and stripping shall not exceed a reasonable underground cost. To determine this cost limit a series of trial sections was made up running normal to a tentative location of the stripping limits. On these sections detailed studies were made showing the ratio of ore to waste and the stripping limit for the particular section determined (see fig. 7 for a typical example of the problems involved). In this study it was necessary to determine the grade of the ore in each section and to ascertain just what amount of stripping could be moved and still show a commercial profit from the ore in question. Referring to the table of stripping ratios, it may be noted that under the costs and conditions upon which the table is based, ore having a grade of 0.82 per cent copper can carry a stripping ratio of 3 to 1 or of 1½ cubic yards of waste to each ton of ore. At 40 cents per cubic yard this means a stripping cost of 60 cents per ton of ore. In some sections the average grade of ore is in excess of 1 per cent copper and as far as grade is concerned could still show a profit for larger stripping ratios permitting the moving of the stripping limit to the increased ratio. But stripping

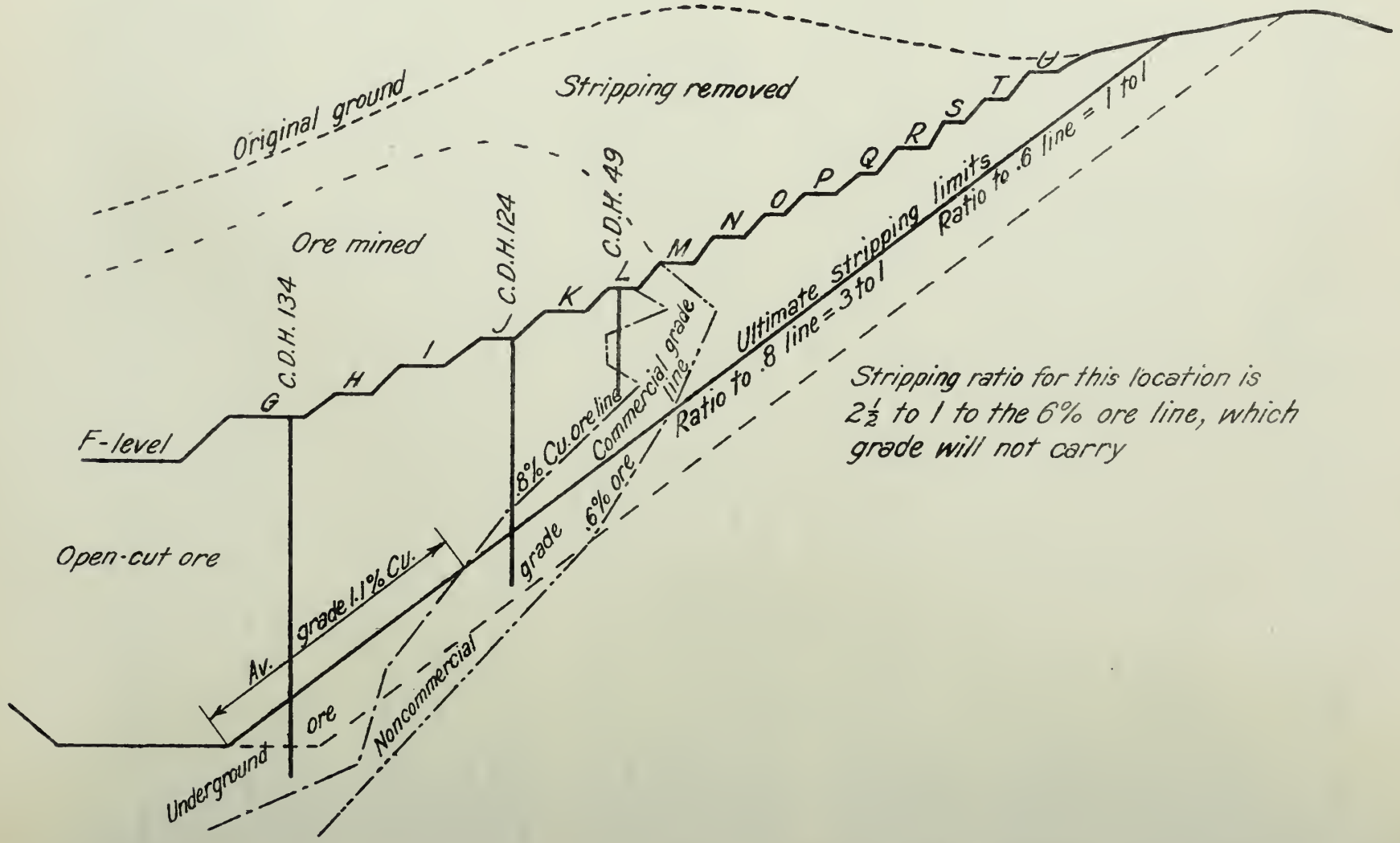


Figure 7. – Typical section showing method of fixing the ultimate stripping limits

costs in excess of 60 cents per ton under the heretofore mentioned conditions, together with other mining costs, exceed a fair underground mining cost; therefore the maximum stripping ratio should not be greater than 3 to 1 and ore outside such limits should be classed as underground ore. Referring again to the section, it may be noted that the .6 per cent ore line extends beyond the stripping limit, but such a grade is not considered profitable by underground mining and is therefore not included in ore reserve calculations. When located within stripping limits, this grade can be classed as commercial ore and included in the reserve.

The study was continued for each section; the ultimate location of stripping limits was then finally laid out, and it was found that of the 625,000,000 tons of ore reserve developed to date, approximately 580,000,000 could be removed by open-cut methods. Possibly some 40 years hence, after open-cut methods have ceased to be profitable, there will be a "mop-up" job to win the remaining tonnage by caving methods. This will entail considerable development work, and plans that are being made for shoveling below the present scene of operations are being laid out to tie in with the possible underground operations.

In addition to the above factors that determine the choice between open-cut and underground methods is the practical side involving the necessity for mass production of a low-grade ore to make it of maximum commercial value. To this must be added the value of flexibility of control of production. To illustrate: If occasion should demand an immediate increase in production to 60,000 tons from a mine ordinarily producing 50,000 tons of ore per day, practically all that is involved in open-cut work is to take two shovels working on stripping and place them on ore. In underground work, to increase the number of ore faces 10 per cent would present a serious problem.

Other issues involved that fix the limit to which open-cut operations can be carried are such factors as maximum degree of railroad curvatures, sufficient space for the efficient operation of power shovels, adjacent property rights, dump grounds for waste material, and above all the safe degree of over-all slope. Calculations of ore tonnages recoverable by open-cut methods and stripping are also dependent upon the slope.

The total ore removed to January 1, 1929, amounts to 175,007,974 tons, having an average grade of 1.21 per cent copper. During the same period 94,338,953 cubic yards of capping and low-grade material has been stripped and dumped in near-by gulches. This gives a stripping ratio to date of 1.1 tons of waste to 1 ton of ore, and is also the ratio being maintained at the present time. The final ratio is entirely dependent upon the ultimate over-all pit slope. Based upon a 40° slope this ratio will be approximately ½ to 1, but, if conditions make it necessary to use a much flatter slope, the ratio may be increased to equal amounts of waste and ore. These are general averages, as there are sections where the ratio reaches the maximum of 3 to 1, and it is in such places that slope is of paramount importance.

SLOPES

When it is considered that in one section of the Utah copper pit the question

arose as to whether or not 38° instead of 40° should be used above a certain level and that the cost of removing this extra amount of waste would reach a sum of $2,000,000, it can readily be seen how vital a thing, at least to the Utah Copper Co., the ultimate over-all slope really is.

The individual bench slope - that is, the actual slope of one shovel face from the upper edge to the toe of the slope - is of less consideration. This is, of course, a function of the over-all slope, but, while an individual bench might stand at 60° from the horizontal, it is not to be concluded that a face 1,500 feet high will stand at the same angle of repose. When the material in any part of the ore body gradually declines in grade from ore to waste, there is always the question as to where the limit of excavation will ultimately be, and it therefore becomes necessary to maintain the shovel terraces on the pit face where, for the time being at least, operations are suspended. This shovel bench can be maintained at a minimum width to accommodate a shovel and loading track--say 30 feet. The slope we are most concerned with is the aggregate made up of the individual bench slopes plus the width of benches. In other words, the over-all slopes would be the angle from the horizontal from the top edge of the excavation to the bottom toe of the excavation, and it is this angle that the Utah Copper Co. has tentatively set at 40°.

There will always be local variations. At some points the slopes will doubtless take the angle of repose of broken material, say 35°. With others a slope of 50° may be safe. Geological conditions will enter into this. For instance, where stripping is being done against a face of quartzite that is dipping toward the operations at 30° from the horizontal, this slope will doubtless turn out to be the dip of the beds. On the contrary, where the beds are dipping into the bench, say from the shovels, a much steeper slope can be maintained.

MINING METHOD

When operations were first started, the initial point of attack was on the west side of the canyon. From the beginning of the first shovel cut, terraces were developed above this cut until there is now a shovel terrace for almost every letter in the alphabet beginning at the yard elevation with bench A and extending up the mountain side to bench W. Sublevels below bench A have also been cut which will be numbered 1, 2, etc., as the down ward excavation of the mine progresses. The actual bench heights vary from 40 to 70 feet as a maximum and bench widths from 30 to 450 feet, the average width being about 100 feet. From the experience of the Utah Copper Co. and under the operating conditions at this mine, a bench height of from 40 to 50 feet is deemed the most economical.

At the mine the elevation of the yard tracks (6,340 feet) practically represents the bottom of the mine as opened at present, and by far the greater part of the ore mined to date has come from above this elevation. The top edge of the open-cut is 1,500 feet vertically above the yard elevation and, since all of the ore from these heights above must come by rail to the assembly yards, a very difficult problem of switchbacks had to be solved. It is, where possible, always desirable to have two entrances to each of the shovel benches; to accomplish this, two sets of switchbacks were constructed, one on what is known

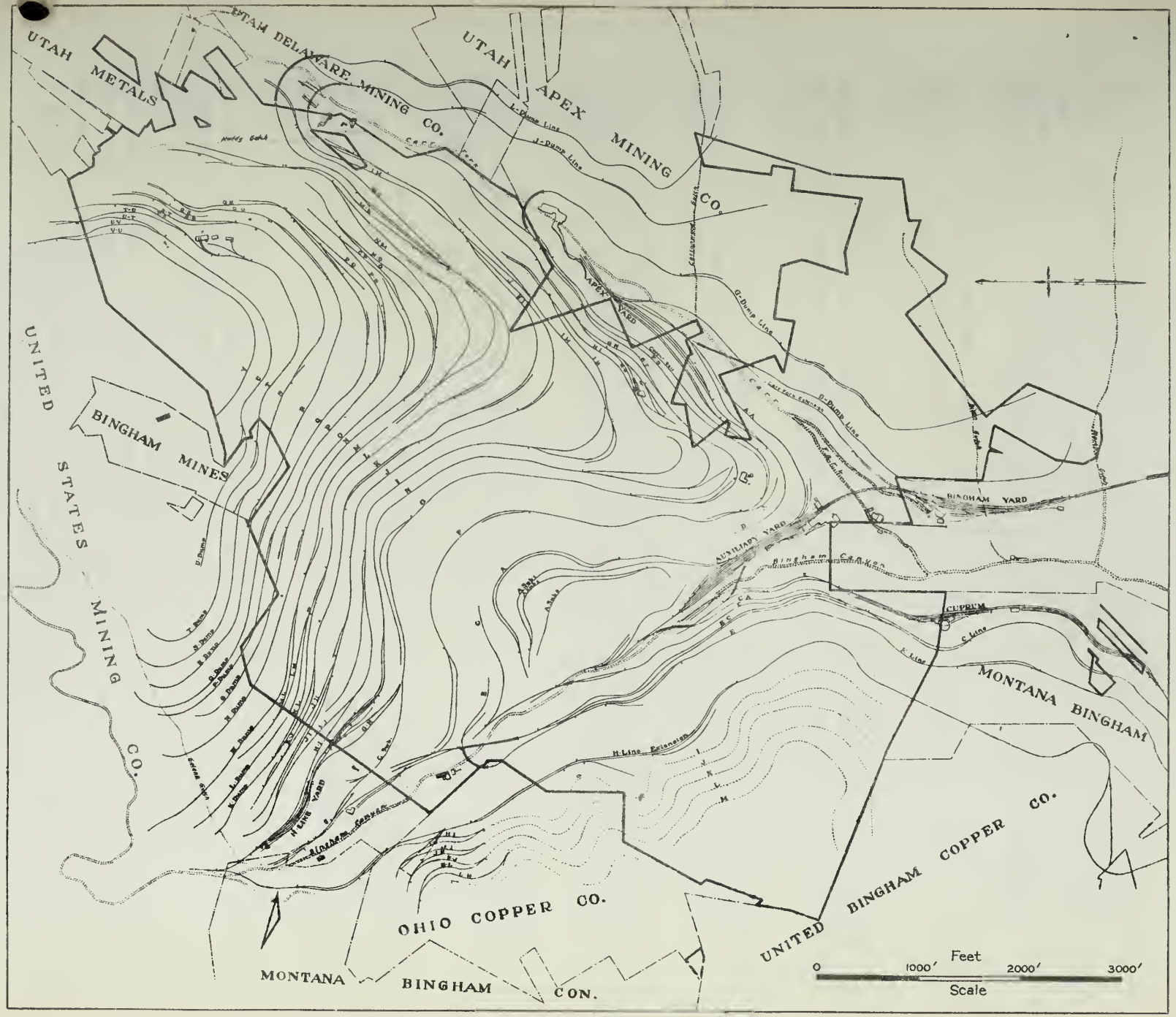


Figure 8. – Utah Copper Company Mine at Bingham Canyon, Utah

as the Carr Fork side of the mine and the other on the opposite or Copperfield side of the mine. The connecting switchbacks are located on maximum 4 per cent grades equated for curvature, and in locating these it was necessary to allow for a tail track of maximum train length. There are approximately 60 miles of trackage in the pits. The distance from the assembly yard to the most remote ore shovel is about 7 miles and requires about 1¼ hours to make a round trip. The average ore haul at the mine is 3 miles (6 miles round trip).

The shovel method in use is the usual power shovel procedure; that is, the ground is drilled and blasted well in advance of the shovel and the trains are loaded in a regular routine way. These trains are dispatched down over the switchbacks by flagmen.

At present the average daily production is 40,000 tons, but when occasion has demanded the production has been increased to 60,000 tons without any unusual stress. As far as shovel equipment is concerned, 90,000 tons a day could be loaded for short periods. In addition to this the present rate of stripping is 30,000 cubic yards per day.

Figure 8 gives a plan of the layout at the Utah Copper Co. mine at Bingham; Figure 3 is typical east-west cross section through the ore and overlying capping, and Figure 4 is a typical cross-section in a northwest direction.

By referring to these sections the terraced arrangement of shovel benches may be noted. These bench heights are not all the same for various reasons. The topography of the ground at the point of entry and the arrangement of switchbacks made it necessary to locate the approach at the most practical point, all things considered.

Shovel operations have been started on the east side of the mine across Bingham Canyon from the main workings. When operations have been completed the excavation will be 2,500 feet deep from the top-most edge to the bottom of the pit. The elongated bowl will be more than 8,000 feet long by 6,000 feet wide.

Waste tracks run out from the various waste levels to nearby gulches, and when the dumps are close to the shovels there is a waste disposal track for each level. Where the waste has to be transported greater distances, one of these disposal lines will serve about three benches, as the benches are connected by switchbacks before reaching the main waste line.

The subject of waste disposal again brings up the question of cut-off between ore and waste. Referring to Table 1, it may be noted that the commercial cut-off is .6 per cent copper under a stripping ratio of 1/2 to 1. When material having a value of .55 per cent copper is within the mining zone, the question arises as to whether it should be sent to the mills or to the waste dumps. Under the conditions upon which the aforementioned table is based, the cost of wasting this material is 20 cents per ton (40 cents per yard). If sent to the mills as ore, the cost would be $1.25 per ton and, as it has a metal content that would yield $1.18, there would be a loss of 7 cents per ton. The conclusion is, that to mine and treat this grade of ore will cost less than to waste it. This example is given to illustrate the point that when horses of waste are encountered within the ore zone, it is the policy of the company, capacity permitting, to mill a small amount of nonpaying material rather than to waste any commercial ore.

In the stripping operations at this mine the comparatively little material that is entirely barren of copper values is limited to the surface capping, which is from 50 to 200 feet thick. The quartzite (which is the country rock of the district), within the stripping limits usually carries values ranging from .2 to .5 per cent copper. The low-grade porphyry, which is also sent to the waste dumps, will average .4 per cent copper. In the early period of operations at Bingham, material up to 1 per cent copper was sent to the dumps, so that the older dumps contain considerable quantities of material that would now be classed as ore. Moreover, it has recently been demonstrated that the copper in such dumps can be very economically recovered.

Copper-precipitation plants have been constructed, and the copper-bearing waters from all the gulches in which dumps have been deposited are conducted to these plants. During the spring run-off, when water is abundant, as much as 50 tons of copper have been precipitated daily on de-tinned scrap. Recoveries are being held at about 98 per cent of the copper content in the water, so eventually the copper in the dumps will be recovered, limited, of course, to the extent to which the waters percolating through the dumps will take copper into solution.

In view of this, a particularly close check on shovel loadings in ore or waste need not be maintained. If a shovel is loading a train of ore cars and runs into small horses of waste (low-grade), it is all loaded and sent to the mills, as it obviously would be an expensive operation to leave it and later load it into waste cars. The same is true for a shovel loading waste cars; small amounts of milling ore are loaded and sent to the dumps, as in time at least a part of the copper in it will be recovered at the precipitation plants.

DRILLING AND BLASTING

When the first shovels were in operation during the summer of 1906, drilling was done by air drills mounted on tripods, and ordinary dynamite was used as the explosive. As the first stripping was done in the vicinity where caving to the surface had resulted from underground mining, it was found that much of the force of the explosion was lost in the broken material. To give a heaving effect to the charge, a portion of black blasting powder was added to the charge. Later, small coyote adits were driven into the toe of the banks for distances of 20 to 40 feet and charges of 10 to 25 tons of explosives were used in the coyote blasts. These large charges interfered with railroad operations because of caving the levels immediately above, so that this system of blasting was abandoned. Toe drilling was again adopted and is the method used at the present time.

The drilling equipment which is used for breaking the ground of the Utah Copper Co. is of very old design and consists of 3½-inch diameter reciprocating drills on tripods operated without the leg weights. Modern drills have been repeatedly tried, but this old-type drill which is used in drilling 22-foot holes still serves better than any that have been recently tried. The fact that the modern hammer machine using hollow drill steel requires water to keep the sludge from in front of the drill bit would make it necessary to maintain water lines on the levels, and this is impossible in winter weather at Bingham.

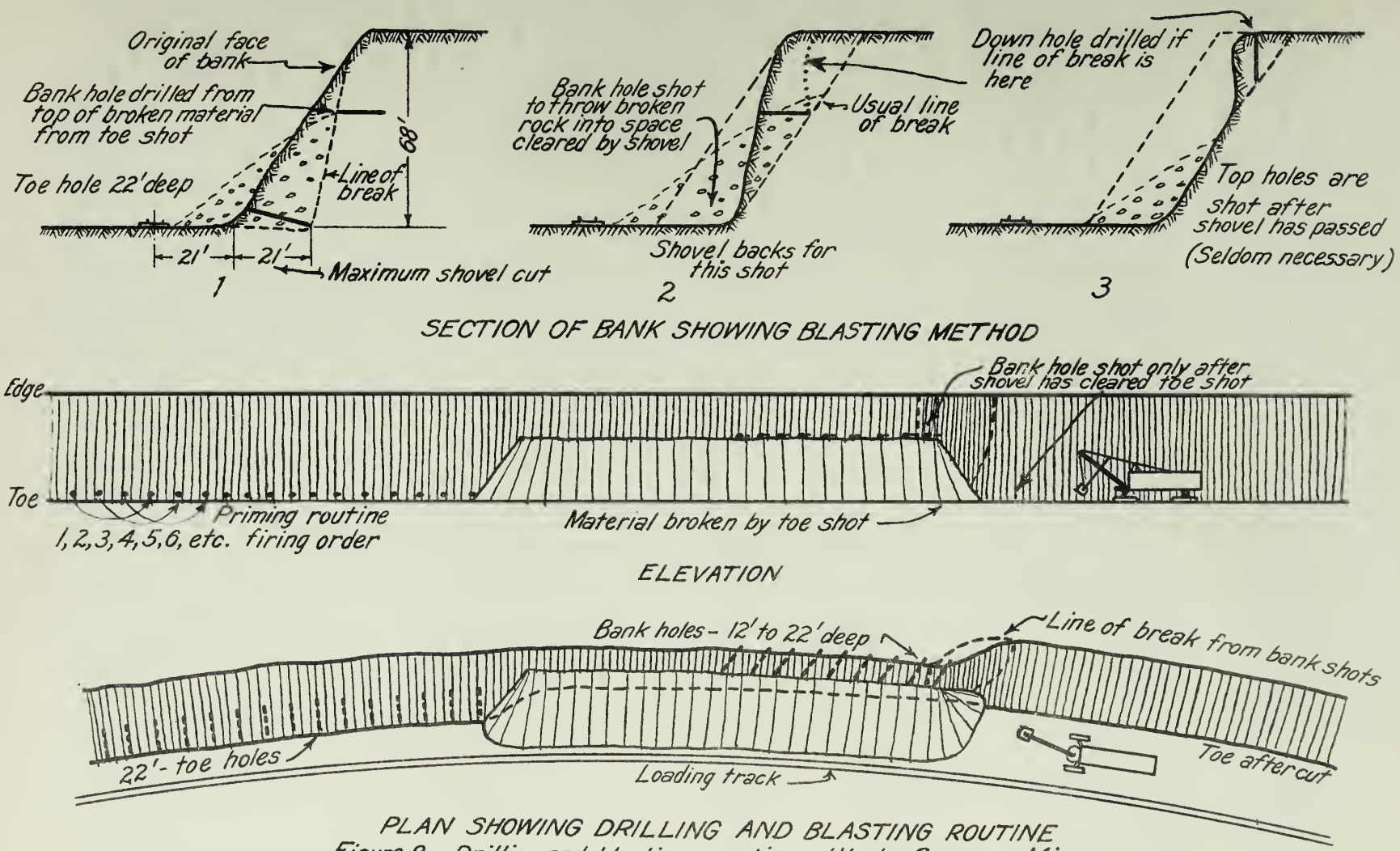


Figure 9. - Drilling and blasting practice Utah Copper Mine

Three compressors furnish 11,200 cubic feet of free air per minute, about 90 per cent of which is used for drilling. The air is supplied to the drills at 75 to 80 pounds pressure through a continuous air line going entirely around the workings and is delivered to each level through headers.

A hole is started with a 3-3/4-inch cross bit and finished with a 2¼-inch bit. The holes are then sprung until a chamber large enough to receive the blasting charge has been made. To chamber the holes usually requires four shots, using approximately 7, 15, 30, and 50 sticks of powder. Water is used for stemming these shots. The main powder charge varies between 150 and 250 pounds. A low-freezing ammonium nitrate powder with a rating of 60 per cent dynamite is used at present. The charge is put in the hole through a 1½-inch pipe and tamped in place. It is fired by means of a primer consisting of a stick of powder having attached to it a No. 6 cap and 6 feet of fuse.

A drilling crew consists of a machine man and two helpers. This crew also does the blasting and trims the bank after the blast when necessary. Two crews are maintained for each shovel, and two general powder foremen have direct charge of the work, one in the upper section of the mine and the other in the lower part. The general foreman tells the crews where to drill and also decides how much powder to use for each hole.

The usual procedure is first to blast the lower part of the bank well in advance of the shovel (fig. 9). The machine is set up on the usual talus 3 to 5 feet above grade and drills downward with an inclination varying from 5 to 15°, so that the hole will bottom at about grade. These holes are from 20 to 25 feet deep and are spaced about 15 feet apart; on the average 18 holes are drilled to the round. It is well at this point to call attention to the fact that the blasting cut should not exceed the reach of the shovel, as it is important that the toe of the cut shall be clean in order to facilitate the drilling of toe holes for the next blast. Another factor that limits the size of the blast is that the broken material must not be of such a quantity as to cover the loading track. A round is fired by three crews who work in tandem. No. 1 crew ignites the fuse and inserts the primer in the first hole, and then proceeds to the fourth hole; No. 2 crew follows the first crew by a few seconds, primes the second hole, and then goes to the fifth; No. 3 crew carries out the same procedure at the third hole then goes to the sixth, and so on until all holes have been primed (fig. 9). A 6-foot fuse gives them about four minutes to complete the operation which, for a round of 18 holes, is ample time. By firing the holes in succession the broken material from the first shot covers the second hole, serving as a blanket for that shot and preventing the material from being thrown over the loading track; and so on with the succeeding shots. After the toe of the bank has been blasted for a distance of 200 feet or more ahead of the shovel, the powder foreman selects those places that the toe shot did not break for the entire bank height in which to drill additional holes about three-fifths the distance up the bank. The holes are drilled horizontally into the bank but at an angle toward the shovel (fig. 9), and are loaded and shot just in advance of the shovel, so that the ore broken by the bank shot falls into the space that the shovel has just previously cleared of the broken ore from the toe blasting. This makes it necessary to back the shovel for the shot, which requires about 5 minutes. Occasionally it is necessary to drill "down-holes" at the top edge of the bank to break off projections of rock that do not break from

the toe and bank shots. Bowlders too large to be handled by the shovel dippers are "dobe" blasted, or block-holed by jackhammers and blasted. After the shovel has quit loading at the end of the shift, the banks are trimmed of loosened rocks near the top to make safe working conditions at the toe of the bank. This is done by hand bars and so-called "pot-holing." Men are let over the top of the bank by ropes and pry down what loose rock they can with hand bars; the rest is shot down by digging small holes 12 to 15 inches deep and loading with a bundle of from 15 to 30 cartridges of powder. This shooting constitutes by far the greater bulk of secondary blasting at the mine.

An average of 45 feet of hole is drilled per drill shift, and 0.022 of a foot is drilled and 1/8-pound of explosive used per ton of material broken. Explosives cost about 1-1/2 cents per ton. Drilling and blasting costs--including labor, equipment, and maintenance of machines and pipe lines--added to the cost of explosives brings the total to a little less than 3 cents per ton. The powder is distributed as follows: 16 per cent is used for chambering, 69 per cent for the main blasts, and 15 per cent for secondary blasting. As has been stated, the ore is a soft porphyry extensively cut up by faults and fissures so that little difficulty is experienced in placing the blasting charges to insure finely-broken rock suitable for shovel-loading. Any difficulty in drilling because of hard ground is also eliminated. A set of drill steel is used for four or five 25-foot holes and then is replaced, not because of losing the cutting edge, but because of loss of gage. Secondary blasting is limited to "blocky" sections of the porphyry or quartzite.

It is the policy of the mine management to have an abundance of broken ground ready in advance of the shovels, as well-broken ground is of the first importance to high shovel efficiency.

LOADING

The first shovels placed in operation at Bingham were 60-ton steam shovels. Soon, however, the 90 to 100 ton railroad-type shovel, on tracks of 4-foot 8½-inch gage, was made the standard digging unit. In 1923 the first caterpillars were placed under these machines, consisting of one tractor under each jack-arm and one trailer tractor under the rear of the cab. This change is without a doubt the greatest advance made in shovel practice during the past 20 years. It meant that no tracks had to be maintained on which to operate the shovels, the short rail sections were dispensed with, and the pit crew was reduced from four or six men, depending on the condition of the shovel floor, to one man. More important than this saving in labor, perhaps, is the facility with which the shovel can by its own power move forward and backward at a moment's notice. It would be difficult to state just how many times a shovel has been saved from being buried by reason of the caterpillar equipment. With the shovels on rails it was necessary to block the machine in position so that it would not move backward while filling the dipper. This, of course, prevented any quick action on the part of shovel crew in getting their machine back out of danger. Every move-up of the machine would require from 10 to 15 minutes, whereas now no time is lost in this part of the procedure. Shovel-loading efficiencies have been advanced materially by the elimination of this delay. Average shovel loadings during 1923, when the old railroad-type steam shovels using 3-1/2-yard dippers were in use, was 2,350 tons of ore per shovel shift, as compared with an average of 3,966 tons for the year 1928 loaded with electric shovels mounted on caterpillars and using 4-1/2-yard dippers.

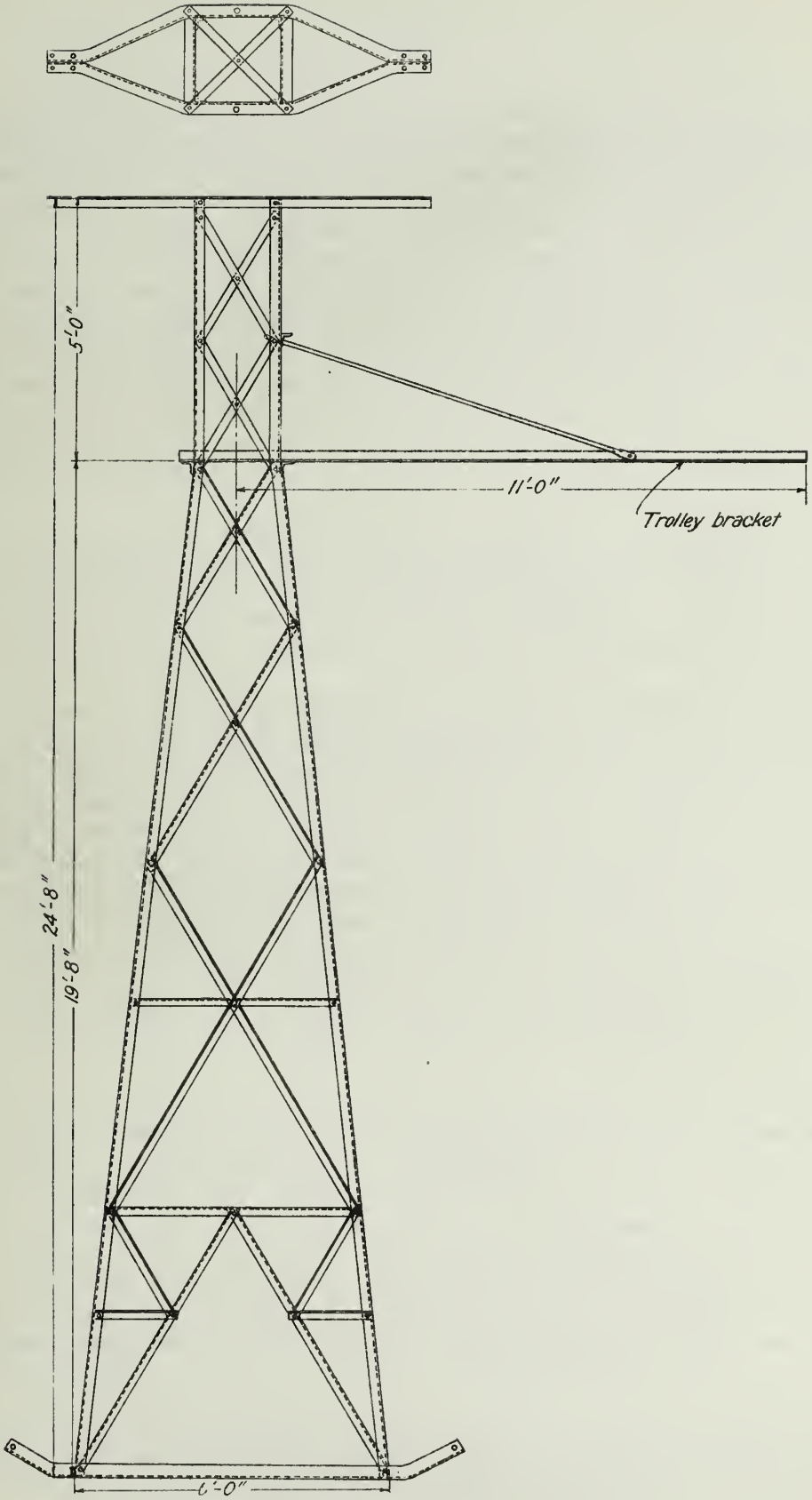


Figure 10.- Portable Transmission and Trolley Tower



In 1923 the first electric shovel was introduced in the mine, which was the same type shovel mounted on three caterpillar tractors as above described. All shovels are now electrified. The crew of these machines consists of the operator, the cranesman, one pitman, and one electrician who looks after two or three machines, depending on the proximity of one to the other. In the change from steam to electric power, the necessary equipment for changing 15 of the old machines was purchased from the manufacturer and installed on the shovels at the company's own shops in Bingham. In addition, eight new shovels were purchased. The savings anticipated as the result of this improvement have been more than realized; many items, such as the servicing of the machines with coal and water, delays due to boiler trouble, and low steam pressures, which, while of real moment, were difficult to set down in cents per ton, have been eliminated by electrification. Regardless of the weather, coal, water, or the vitality of operator, the machine is ready at full operating pressure the moment the switch is closed.

There are 23 electric shovels in service at the mine; 8 are of the alternating-current and 15 of the direct-current type. The alternating-current shovels have rheostatic control and the direct-current shovels use the Ward-Leonard control. The booms are 30 feet long and swing through an arc of 190°, which makes it possible for the shovel to make a 20-foot cut across the level. The dippers have a capacity of 4-1/2 cubic yards.

Electric power for the Utah Copper Co. operations at both the mine and mills is purchased from the local power company and is transformed to 44,000 volts at the company's central station at Magma. Two three-phase lines of No. 2 copper wire, 14 miles long, extend from Magma to Bingham, where at a central switch rack they split into a two-circuit line. These feed lines supply two outdoor substations, each of which contains six 400-k.v.a. transformers where the tension is reduced to 5,500 volts for shovel operations. The power from each substation is carried over two single-circuit lines, mounted on wood poles, which encircle the entire mining area. These lines are built close to the outlets of the different mine levels, and with this system of distribution it is possible to feed any level from any one of the four secondary trunk lines. The reason for such flexibility in the secondary distribution system is to provide insurance against delays, as continuity of power supply is of utmost importance to uninterrupted shovel performance.

The problem of getting power from the distribution lines to the shovels was a difficult one because of the danger of blasting the lines, and further because such power lines must be portable. Rubber-covered cable insulated for 7,500 volts was tried, but was supplanted by light, portable steel towers mounted on skids (fig. 10). These towers are approximately 25 feet high and 6 feet square at the base. They carry three No. 2 stranded copper wires in average spans of 125 feet. Each shovel receives power from this portable line through a connecting 500-foot length of four-conductor trail cable, three of these conductors being for power and the fourth for ground. This cable is wound on a reel mounted on the rear of each shovel and permits moving the shovel a distance of 400 feet before the connection must be changed.

The general plan for loading ore at the mine is to work full capacity during the daylight shift. At the end of the day shift the amount of ore loaded is computed, and enough shovels are detailed for loading ore on the night shift to make up the quantity re-

quired for that particular day. The night shift is also utilized for maintaining good working conditions at the waste dumps. When one of the dumps in a series is lagging behind the others, stripping is removed on the corresponding mine level at night and added to this dump.

A shovel will start making its cut at one end of a bench and will proceed across the bench to the other end, loading the material whether it be ore or waste. When it reaches the end of its travel, the operation is repeated. As the different shovels move across their respective benches, their progress is so arranged that one shovel will not be directly above another.

Usually ten to twelve 80-ton ore cars are brought in and spotted for loading, the locomotive remaining coupled. When the train is loaded and enroute to the yard, another locomotive will bring in the next train, and the process of loading continues. The procedure is the same in loading waste. The average capacity of the shovels is about 600 tons per hour, and an ore train with a capacity of from 800 to 960 tons will require from 1½ to 1¼ hours to load. Waste trains consist of from three to seven 30-yard cars, depending on the proximity of the waste dumps.

The following table gives the performance of the shovels for 1928:

Table 3.- Performance of Shovels

Type of shovel	Material	Shifts worked	Loaded	
			Tons	Tons per shift
A.c.	Ore	929.87	3,613,720	3886
A.c.	Waste	1651.76	6,908,460	3692
	Total	2581.63	9,712,180	3762
D.c.	Ore	3244.74	12,942,350	3989
D.c.	Waste	2228.62	8,301,710	3725
	Total	5473.36	21,244,060	3881
A.c.	Ore	929.87	3,613,720	3886
D.c.	Ore	3244.74	12,942,350	3989
	Total	4174.61	16,556,070	3966
A.c.	Waste	1651.76	6,098,460	3692
D.c.	Waste	2228.62	8,301,710	3725
	Total	3880.38	14,400,170	3711
	Grand Total and Average	8054.99	30,956,240	3843

The table shows that both the alternating-current and the direct-current shovels

loaded more per shift in ore than in waste, the average of the two being about 7 per cent more. It also shows a slightly higher performance of the direct-current over the alternating-current shovels. During the good-weather months the shovels loaded about 7½ per cent more than the average for the year. In 1928, from May to October, inclusive, the alternating-current shovels averaged 4,031 tons per shift as against 3,762 tons, the average for the year, while during the same months the direct-current daily average was 4,097 tons as compared with 3,881. Loadings as high as 7,000 tons have been made in a shovel shift.

The direct-current shovel will load 25 per cent more material than the alternating-current shovel with the same power, as shown by the following data for 1928. The cost for power was 15 per cent of the total cost of shovel operation.

	Alternating-current shovels	Direct-current shovels	Totals
Kilowatt-hours	2,380,041	4,202,539	6,582,580
Tons loaded	9,712,180	21,244,060	30,956,240
Tons per kilowatt-hour	4.081	5.055	4.703

During 1928 the shovels were actually loading 82.1 per cent of the total time they were on shift. The loss of time from electrical trouble is the smallest of all of the losses, and the most time lost is due to switching ore or waste cars. This loss of time is inherent in the method and could be avoided only by having passing tracks extending the full length of the levels so that an empty train could be spotted immediately upon the departure of the loaded train. It is, however, impracticable to have more than one track across a level.

A complete description of the shovel electrification appears in the September, 1927, issue of the Mining Congress Journal in an article written by C. W. and R. J. Corfield, electrical engineers for the Utah Copper Co.

TRANSPORTATION

Transportation equipment has shown marked change in size since the beginning of the Utah Copper Co. Twenty-ton locomotives, 6 cubic yard dump cars, and 30-pound rails have long since been abandoned. Prior to the recent electrification of the haulage at Bingham the steam locomotive weighed 85 tons and was modern in all respects, even to superheaters, etc.

The success in electrifying the shovels served to speed up the electrification of mine haulage, so that at this writing forty-one 75-ton electric locomotives are in service that have replaced a like number of steam units. The question as to whether or not the storage-battery type of locomotive should be used or whether a trolley should be strung along all the operating faces for a trolley-type locomotive was a matter of long consideration. Two experimental units were purchased, one a Deisel-electric 60-ton engine, and the other a combination trolley-storage battery locomotive. The result has been an almost exclusive use of the trolley type. For emergency purposes, seven locomotives of the combina-

tion type were selected which can be operated independently of the trolley. All locomotives are so constructed as to permit ballasting up to 90-tons. They are of the articulated type, and are capable of hauling 12 empty ore cars, each weighing 21 tons, up the 4 per cent switchbacks at 12 miles per hour.

Due to the complexity of the trackage system, it was necessary to provide for different methods of collecting power from the trolley, as wires can not always be placed over the center line of the track. Because of this the locomotives are equipped with a conventional main pantograph, two side-arm collectors and a cable reel containing 2,000 feet of cable. The pantograph and side-arm collectors are air-operated.

To date, approximately 40 miles of trolley have been erected, standard catenary construction being used for trolley line over permanent tracks and direct suspension else where. Due to the necessity of operating spreaders to clear snow from the tracks, all poles are placed on the bank side of the track. On the working levels the steel towers used for the feeder lines to the shovels are also used to support the trolley, a standard bracket arm being attached to the tower for this purpose (fig. 10). These towers must be moved with each track change. Where dump tracks cross large fills a special type of construction is required because the dumps are continually settling. In such places the trolley support is made a part of the track to maintain a fixed clearance between rail and trolley wire.

In line with the company's policy of continually improving operating equipment, waste cars have been gradually increased in size until the 30-cubic-yard unit is now standard. In the matter of ore transportation the capacity of the cars has been 80 tons, but the new equipment recently put into service has been increased to a capacity of 90 tons on a running gear of 100-ton capacity. As ore tonnages have increased, the 65-pound rails first used for the mine track system have been replaced by 90-pound rails on the switchbacks, and the 65-pound rails thus released are used on the shovel levels and dump tracks. As replacements and new construction on the levels are made necessary, 90-pound rails are used.

Each shovel, whether loading waste or ore, is served by two locomotives. With two exceptions, all the switchbacks are single track, and the movement of the trains on them is controlled by flagmen placed at advantageous points. During cloudy or snowy weather, when it is impossible to employ visual signals, a telephone system is used which has been installed exclusively for this purpose. Whenever a train is ready to leave a bench the engineer asks for a signal by a whistle. If permission to leave is given by the flagman in charge of that section, he proceeds to the next flagman where he again has to whistle for signals. The mine locomotives take the ore cars to a main assembly yard where they are made up in 50-car trains and hauled to the mills by 320-ton Mallet locomotives.

In disposing of stripping in the gulches, the first dumping is at the end of the gulch farthest from the shovel bench. As this part of the dump is filled, the waste is dumped successively closer to the shovel bench until the entire length of the dump for that position of the track has been filled. The dump gang then levels off the dump and shifts the track to the edge. One dump gang, consisting of a boss and approximately 20 men, takes care of more than one dump where each dump is serving but one shovel. Where trains from two or more shovels run to one dump and dumps are high, the entire services of one gang may be required for the one dump.

Track shifting on the shovel levels is done by Petersen Track Shifters, of which there are nine in service. One of these machines requires a crew of six men to move the track. Three to five track gangs of 20 men each are also used in this work to line-up the track, etc. Track shifting on the dumps is done by track gangs using lining bars. Track shifters are used on the dumps only where the settling of the dump necessitates raising the track to grade.

DRAINAGE

Drainage at the mine is for the present a comparatively simple problem and requires no pumping. All mine levels are located on the side of the mountain, with the exception of the lowest level at the bottom of the pit. Water that drains into this level from the upper parts of the workings is passed through a churn drill hole in the pit to the Mascotte Tunnel which extends through the mountain on the east side of Bingham Canyon to Salt Lake Valley. Water from the higher levels is taken care of by the drainage system used for the town of Bingham, situated in the bottom of the canyon.

PLANT

Completely equipped machine, drill, blacksmith, carpenter, locomotive, and shovel repair shops are maintained at the mine and are located at the assembly yards. A mine warehouse is also located at the main assembly yard and supplies are delivered to different parts of the operation by a tramp engine and flat car. Water supply tanks are located at points of vantage outside the mine, from which water lines are run to all parts of the operation. A new compressor plant has been recently installed because the old plant was located too close to the mine and within the stripping limits. Air lines and power lines have been previously described. The water supply for the mine is taken from Middle Canyon at an elevation high enough to make possible distribution by gravity. No pumping is necessary.

PER CENT EXTRACTION

It can safely be said that the ore extracted from the Utah Copper mine to date is 100 per cent of the total in the area excavated. Some engineers might question this, but with the erecting of precipitation plants, copper existing in the waste dumps is being recovered. It has been said at the mine since the installation of these plants, that "We have no waste, but low and lower grade ore."

As stated previously, about 85 per cent of the total ore will be recovered by shovels and the remaining 15 per cent will be mined by underground methods which will probably not recover more than 90 per cent of it.

WAGES, CONTRACTS, AND BONUS SYSTEM EMPLOYED

The Utah Copper Co. mine is operated on a straight wage and salary basis, and no bonuses of any kind are paid. Isolated jobs are often done by contract, and the construction

of trestles, houses, etc., is almost invariably given to the lowest bidder on a contract basis. With a production of 60,000 tons of ore daily, approximately 2,000 men are employed exclusive of railroad and mill hands.

SAFETY METHODS AND FIRST-AID ORGANIZATION AND TRAINING

The purpose of the safety department may be summed up under three headings:

1. Investigation of all accidents

All accidents, no matter how trivial, are thoroughly investigated and the causes determined so that the necessary changes in equipment, methods, and placing of men can be made to prevent their reoccurrence.

2. Distribution of safety propaganda

Safety educational activities include the maintaining of 52 widely distributed Bulletin boards upon which bulletins are changed weekly, safety campaigns, no-accident periods, etc., 100 per cent first-aid training, employees' safety publications, etc.

3. Safety inspections

Regular inspections are made of all equipment, buildings, methods of doing work, working places, etc., by a corps of safety inspectors.

The hill is divided into three sections, and a safety inspector assigned to each section. The inspector is responsible to the safety engineer and investigates all accidents that occur in his section. He also submits a written weekly inspection report to the safety engineer covering all conditions in his section.

Semimonthly meetings of all department heads are held in the superintendent's office, the superintendent acting as chairman. The report and recommendations of the safety engineer, the accidents which occurred during the previous two weeks and any other matters pertaining to safety and welfare are discussed at this meeting.

A monthly meeting of all foremen and bosses is presided over by the safety engineer. Although all accidents of the previous month are discussed and some first-aid training is given, the chief purpose of this meeting is to create good fellowship and morale among the foremen.

The following tabulation shows the extent to which accidents have been reduced at the mine by the foregoing program:

Frequency rate	1919	1920	1921	1922	1923	1924	1925	1926	1927	1928
Number of accidents per 1,000 men per month	23.6	22.3	16.5	29.1	19.8	20.5	18.0	18.5	6.0	3.7
Severity rate										
Days lost per 1,000 man hours	10.1	9.8	1.16	10.1	10.5	7.4	6.1	8.7	3.5	2.9

FORM OF ADMINISTRATIVE ORGANIZATION

The administrative and mine organization of the Utah Copper Co. is shown in detail in the accompanying organization chart (fig. 11).

Table 4.- Summary of Costs

Utah Copper Co. Mine — Year of 1928

Cubic yards waste loaded during period - 7,220,034

Operating Cost per cubic yard of stripping

	(1)	(2)	(3)	(4)	(5)	(6)
Operating labor	$.0207	$.0128	$.0238	$.0565	-	-
Air	.0033	-	-	-	-	-
Explosives	.0337	-	-	-	-	-
Other operating supplies	.0006	.0007	.0006	.0142	-	-
Repair labor and supplies	.0021	.0197	.0226	-	-	-
Power	-	.0064	-	-	-	-
Bench transmission lines	-	.0020	-	-	-	-
Fuel	-	-	.0172	-	-	-
Dump cars	-	-	.0103	-	-	-
Supervision	-	-	-	-	.0132	-
General	-	-	-	-	.0133	-
Totals	$.0604	$.0416	$.0745	$.0707	$.0265	$.2737

Dry tons ore loaded during period: 16,558,500.

Direct operating cost per dry ton of ore

	(1) Drilling and blasting	(2) Loading	(3) Transportation	(4) Track maintenance	(5) Other mine charges	(6) Total
Operating labor	$.0094	$.0056	$.0104	$.0124	-	-
Air	.0015	-	-	-	-	-
Explosives	.0151	-	-	-	-	-
Other operating supplies	.0003	.0003	.0003	.0045	-	-
Repair labor and supplies	.0009	.0090	.0097	-	-	-
Power	-	.0031	.0003	-	-	-
Bench transmission lines	-	.0008	-	-	-	-
Fuel	-	-	.0074	-	-	-
Yard and train	-	-	.0080	-	-	-
Ore cars	-	-	.0006	-	-	-
Supervision	-	-	-	-	$.0058	-
Churn drilling	-	-	-	-	.0058	-
General	-	-	-	-	.0061	-
Totals	$.0272	$.0188	$.0367	$.0169	$.0177	$.1173

Table 5.- Summary of costs in units of labor, power, and supplies

Utah Copper Company Mine — Period - October, 1929

Material loaded during period:

Total material, cubic yard - 1,343,579
Ore, dry tons ..... - 1,249,500
Waste, cubic yard ..... - 741,990

	Stripping, per cubic yard	Mining, per dry ton	Total, units per ton ore
A - Labor (in man hours):			
Drilling and blasting	.032	.014	.033
Loading	.028	.011	.028
Haulage	.057	.024	.058
Track maintenance	.122	.031	.104
Repairs	.084	.025	.074
Miscellaneous	.004	.003	.005
Supervision	.024	.010	.025
Total labor	.351	.118	.327
Average yard or tons per man per shift	22.7	67.9	24.5
Labor, per cent of total cost[1]	63.6	56.9	60.3
B - Power and supplies:			
Explosives (lbs.)[2]	.241	.105	.248
Total power (kw.h.)-			
1. Shovels	.42	.20	.46
2. Locomotives	.50	.36	.66
3. Air compressors	.53	.23	.54
4. Shops	-	-	.04
5. Lighting and miscellaneous	-	-	.14
Total power	1.45	.79	1.84
Fuel (tons of coal)	.003	-	.002
Other supplies in per cent of total power and supplies[1]	58.0	49.5	53.8
Power and supplies, per cent of total cost[1]	36.4	43.1	39.7
C - Percent of total cost	100.0	100.0	100.0

1 - Based on ten months of 1929

2 - Blasting powder: Ammonium nitrate, permissible type, low freezing, rated equivalent to 60 per cent dynamite

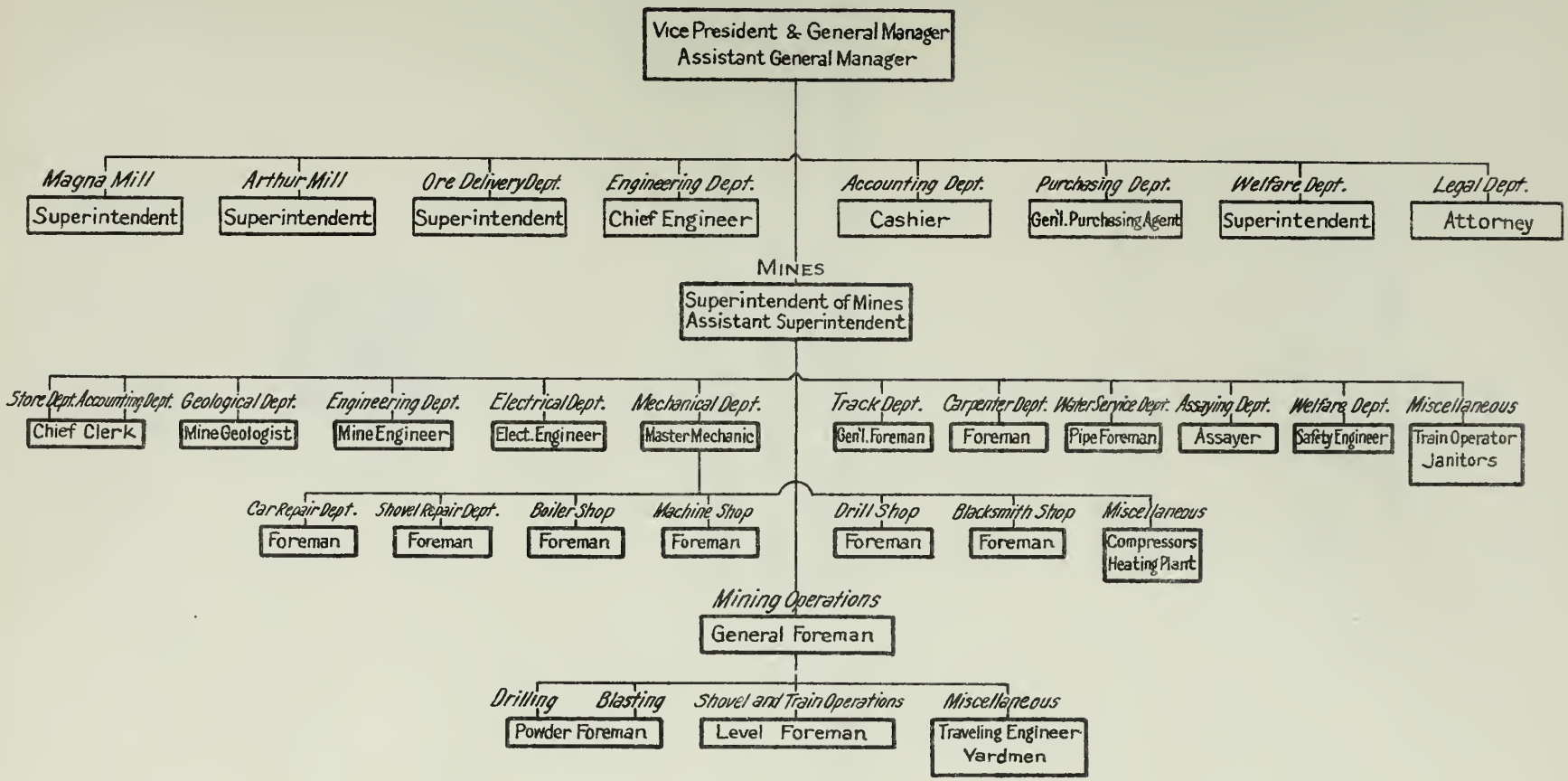


Figure 11.- Organization Chart

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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

COMPARATIVE ADVANTAGES OF APPLYING SEVERAL GEOPHYSICAL METHODS OF PROSPECTING TO THE SAME TERRITORY

BY

F. W. LEE

I.C.6235.
February, 1930.

INFORMATION CIRCULAR

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COMPARATIVE ADVANTAGES OF APPLYING SEVERAL GEOPHYSICAL METHODS OF PROSPECTING TO THE SAME TERRITORY[1]

By F. W. Lee[2]

INTRODUCTION

In the gradual development of the science of applied geophysics there have been many schools of thought, and each approaches this difficult problem from a different angle. Some of the practices advocated had their origin in mere speculation and psychological fantasies; others had sound premises based upon physical laws. The former have largely been abandoned, whereas further tests and research have continuously strengthened the latter.

During this evolutionary period it was only natural that certain methods should have a wider appeal than others. Consequently attention was focused principally on the more popular methods to the exclusion of others and this interest was often augmented by much publicity. In some instances overzealous promotion extended the application of these methods beyond their useful limits and possibilities, resulting in inevitable failures that tended to discredit the methods involved. Another complicating factor was the development of many independent methods, some based upon the same and others upon different physical reactions. Often a mine operator was at a loss to know which method to use. His choice in many instances was decided by the results obtained with various methods in other localities rather than by the applicability of a given method to his particular problem. That all geophysical prospecting methods have certain inherent restrictions depending upon many diverse factors is admitted by everyone. The purpose of this paper is to point out what some of these restrictions are, how they exhibit themselves, and how further light may be shed upon them.

ACKNOWLEDGMENTS

The results of this work were accomplished under the joint auspices of the Canadian Geological Survey and the U S. Bureau of Mines at the Falconbridge and Errington properties, located in the Sudbury Basin of the Province of Ontario, Canada.

The writer wishes to thank Dr. A. S. Eve, Dr. D. A. Keys, Dr. G. Gilchrist, Dr.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6235."

2 - Senior physicist, U. S. Bureau of Mines.

J. B. Mawdsley, and Mr. W. G. I. Watson of the Canadian Geological Survey for their cooperation in the work incident to this report. The assistance of Mr. J. Wallace Joyce in preparation of the manuscript is also acknowledged.

The generosity and aid of Mr. E. Craig and Mr. W. Samuel, at Falconbridge were invaluable. Equal generosity was shown at Errington by Mr. V. C. Clauson and Mr. E. L. Stenger.

EVALUATION OF RESULTS

All mining operators who employ geophysical prospectors are entitled to know how much dependence can be placed on the interpretation of geophysical measurements. In such evaluation it is taken for granted that all measurements have been accurately made, with due allowance for all corrections and errors that pertain to the method of prospecting. The resulting final interpretations are those that will guide the operator in making his decisions as to further procedure.

To illustrate how various factors govern the value of some geophysical exploration methods, two typical examples of work in the field will be considered. Both of these examples are taken from explorations in the Sudbury Basin made in cooperation with the Geological Survey of Canada. The work was done on the Falconbridge and Errington properties, which are about 25 miles apart. Both properties had been thoroughly examined by diamond drilling and mine exploration, leaving little to speculation.

FALCONBRIDGE DISTRICT

The ore deposit consists of a massive sulphide contact deposit running east to west and dipping slightly to the north. The ores (pyrrhotite, pentlandite, and chalcopyrite in various proportions) are valuable for both their nickel and copper content. Glaciated sands and bowlders cover the ore body for a depth of 92 to 117 feet. Figure 1 shows a plan of the ore body and Figure 2 a vertical cross section. The ore body is well defined on the north by an intrusive mass of norite and on the south by quartzite-graywacke beds. A more complete geological description by Dr. Mawdsley of the Canadian Geological Survey is given in the appendix.

Factors That Determine Choice of Methods in This District

Magnetic Surveys.- As all of the ores contain much iron, magnetic disturbance should be expected. Hence, a careful survey of the vertical component of the earth's field by a good magnetometer should give some information. Controverting this expectation is the possibility of a wide distribution of pyrite over a considerable region, which would cause anomalies that might overshadow the effect of the ore body. On the other hand, the amount of iron in the deposit might not suffice to give a greater indication than the normal earth anomalies. The effect would be the same if the iron in the mineral had a low susceptibility, if, for example, the mineral had the magnetic susceptibility of serpentine. Thomas A. Edison has made extensive surveys here with the magnetic needle. He reports that the attraction was very small and that he took little interest in the results. A shaft was sunk under his direction at the point of greatest attraction until quicksand and water was struck, then it was abandoned. Later developments disclosed that the shaft stopped within a few feet of the ore.

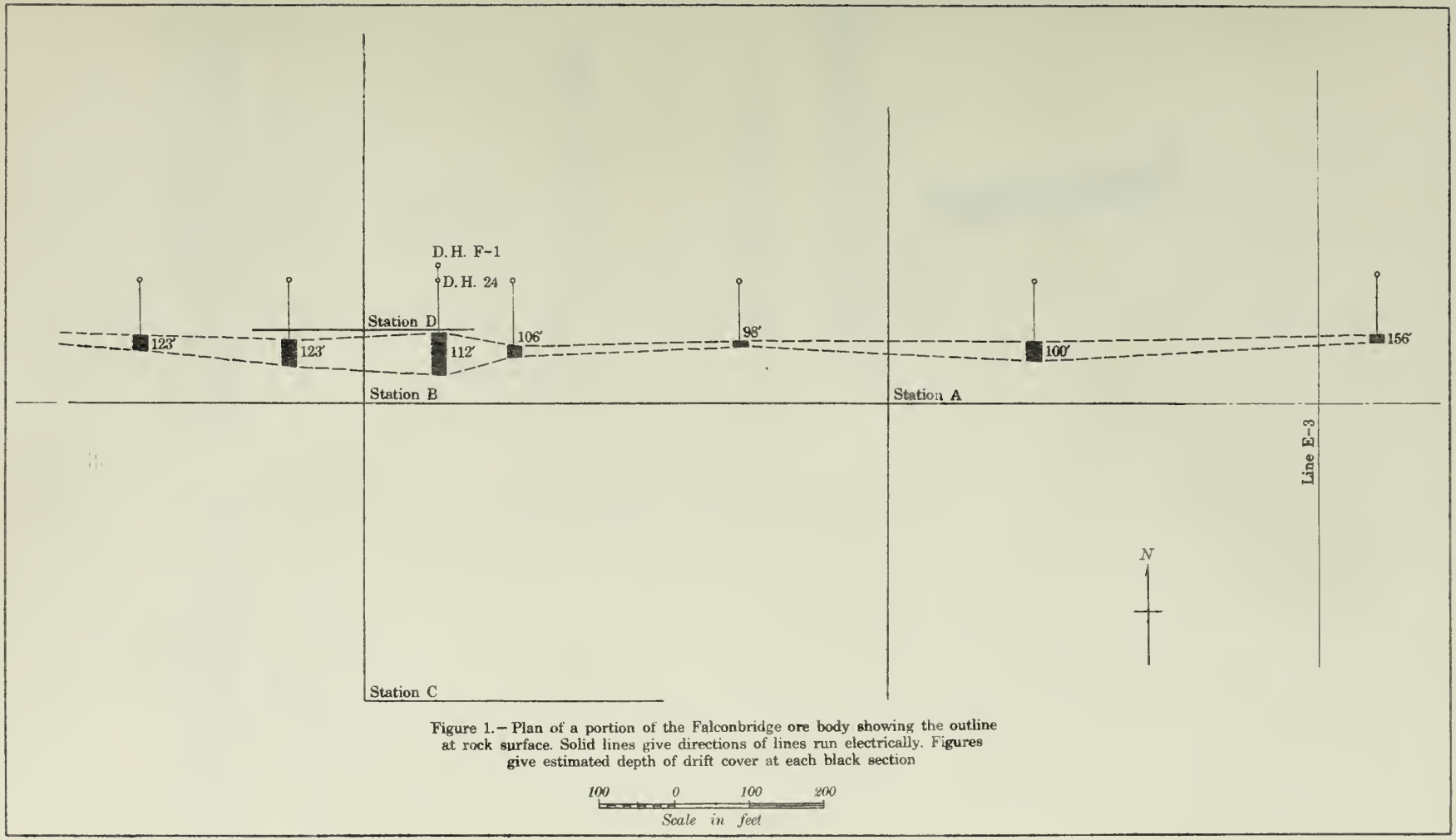


Figure 1.– Plan of a portion of the Falconbridge ore body showing the outline at rock surface. Solid lines give directions of lines run electrically. Figures give estimated depth of drift cover at each black section

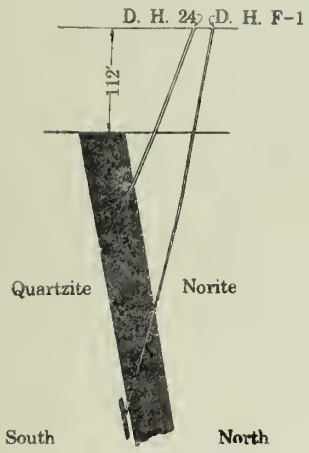


Figure 2. – Vertical cross section of the Falconbridge ore body taken in north and south plane through diamond-drill holes 24 and F–1, showing thickness of drift cover and position of ore zone

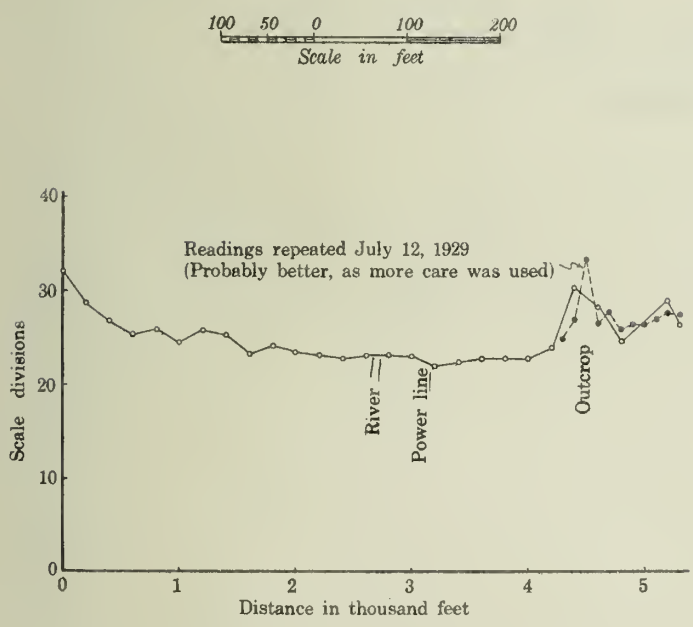


Figure 3. – Magnetic survey with Askania vertical magnetometer on barren ground in the Falconbridge district. Measurements taken on main E–W line to E–3 line and thence south across marsh, July 11, 1929

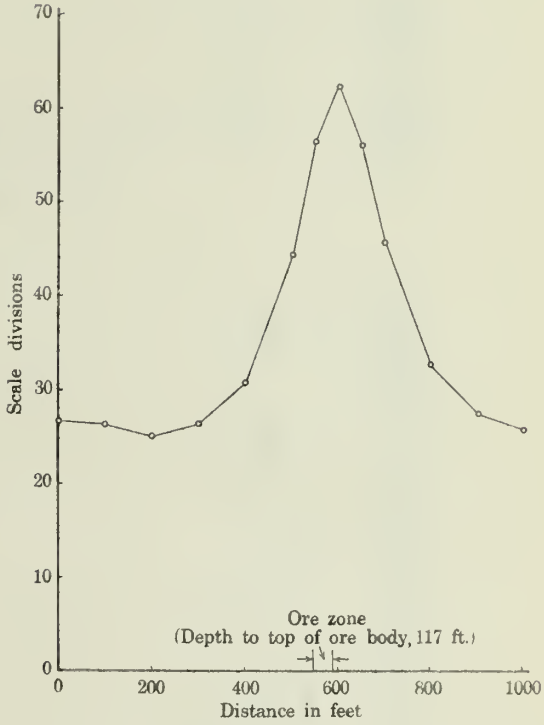


Figure 4. – Magnetic observations across the Falconbridge ore body with an Askania vertical magnetometer, going north from a point 100 feet south of station C, July 9, 1929

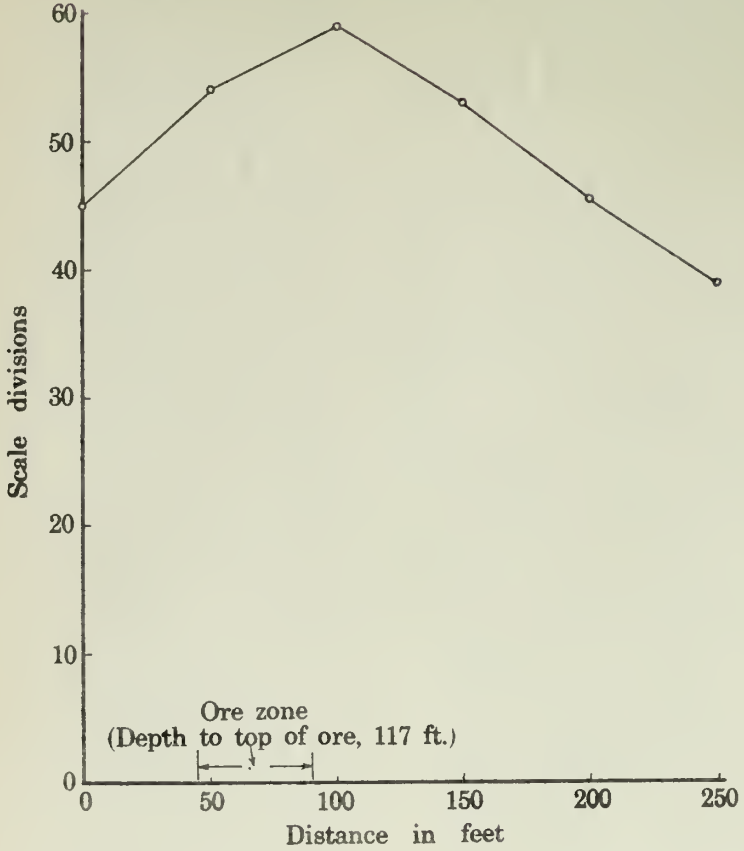


Figure 5. – Magnetic observations across the Falconbridge ore body with an Askania vertical magnetometer, going north from E–W line 60 feet east of station B, July 6, 1929

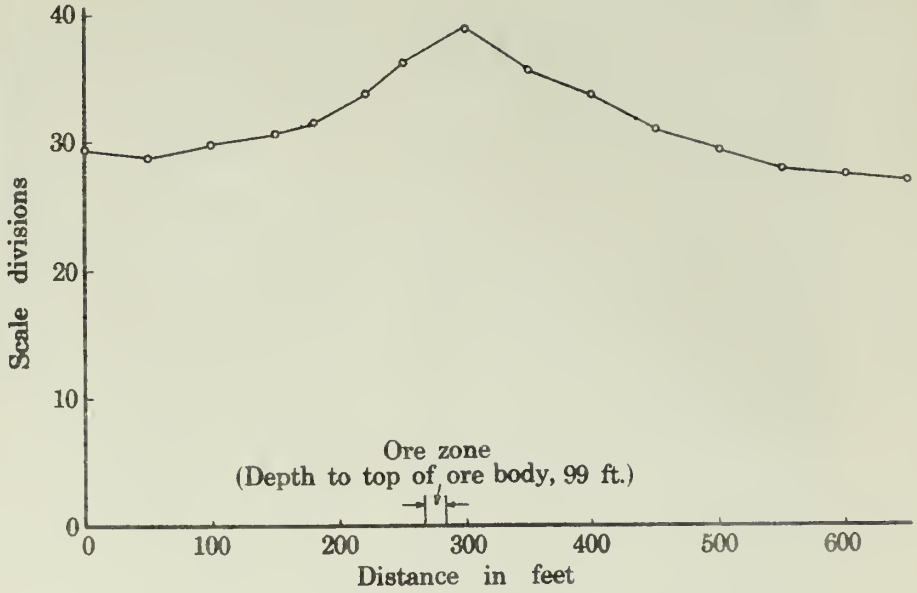


Figure 6. – Magnetic observations across the Falconbridge ore body with Askania vertical magnetometer, going north from a point 200 feet south of survey post at corner of lots 10 and 11 and concessions 3 and 4, July 6, 1929

Self-Potential Surveys.- Because the ores are sulphides they can be oxidized by air dissolved in water percolating through the sand, and the resulting chemical action causes a potential and current field around the ore body. However, sand formations are very porous, and hence are not likely to hold percolating waters containing dissolved salts which would occasion the production of electric cells due to diffusion of solutions of various concentrations. Furthermore, as the sand is a uniformly poor conductor it would allow the electric field to spread to a great distance and would not short-circuit it as a good conductor would. Moreover, the great depth (over 100 feet) to which the surface water has seeped and the great distance at which the current must be observed are complicating factors. Although the natural surface potentials in a glaciated district of this character may be small and apparently favor accurate results, one disturbing element, depth, will counteract favorable surface conditions. Other pyritic deposits of noncommercial value may also be oxidizing in a similar way and generating a potential.

Electrical Resistivity Surveys.- Because the ore itself has a resistivity of about 8,000 ohms per centimeter cubed and is a good conductor compared to the overburden with a resistivity of more than 1,000,000 ohms per centimeter cubed and compared to the adjacent rocks which also have a high resistivity, one may locate the ore body by a resistivity method. However, it is difficult to apply enough current to the ground, due to the high resistance of the sand, to allow accurate measurement with a potentiometer reading to millivolts. There also may be other conductors below, which may interfere with the discovery of those desired.

Results of Magnetic Surveys

At first, various measurements were made across the ore body at different places, but the significance of these measurements could not be ascertained until a general magnetic survey was conducted in barren territory. Figure 3 gives a survey of the barren ground. The measurements were made with an Askania vertical magnetometer, each scale deflection representing about 36.38 gammas. From this diagram one can see that the vertical magnetic intensity is uniform in the district, except where a small rise occurs at 4,500 feet, as shown on the diagram. By comparing this relatively uniform field with the measurements made across the ore body, shown in Figures 4, 5, and 6, a marked difference is apparent which indicates the presence of the ore body. Variation in the scale of plotting causes some of these curves to appear broader than others. There is a maximum variation of about 37 scale readings, equivalent to about 1,343 gammas, which is approximately 0.3 per cent of the normal vertical earth's field of about 400,000 gammas.

It is of equal importance for a company to have a record of curves such as are given in Figures 3, 4, 5, and 6 as it is to have the location of the ore body as determined by geophysical investigations on the map of the territory. An interesting feature of these curves is that the maximum variation is not directly over the ore body or toward the south, as might be expected from its dip, but toward the north. This may indicate an enlargement of the deposit at greater depth to the north. The same effect is also indicated in Figures 4 and 5. The rapidity, ease, and accuracy with which these magnetometer measurements can be taken, requiring about seven minutes to a station, makes the method well suited for reconnaissance in this locality. Stations should not be more than 200 and preferably 100 feet

apart so that the position of an ore body will not be overlooked. It should be remembered, however, that if the normal ground caused the same variation as that produced by the ore body this method would have no value.

Results From Self-Potential Methods

With the aid of a potentiometer and porous pots a potential survey was made by measuring the potential difference over every 100-foot interval. Although the top of the ore body was 100 feet beneath the surface, its effect at the surface was nevertheless noticeable, undoubtedly due to the very favorable overburden of porous sand of high resistivity. A maximum value of 63 millivolts was observed as compared with the usual variations in normal barren ground of 10 to 15 millivolts. This method indicated that the ore deposit, as seen in Figures 7 and 8, was in exactly the same place as the magnetometer observations indicated. This fact is especially well shown in Figure 7, but is not quite as accurately indicated in Figure 8, where the reaction is smaller and the normal ground potentials overshadow the potential field from the ore body.

Resistivity Measurements

To check the previous results by further methods operating independently of the other two systems, resistivity surveys were made. These surveys were of two types; in one measurements were made around a single electrode, and in the other, two electrodes were used in a modification of the Gish-Rooney method.

Where measurements are taken around a single electrode the other electrode is moved so far away that the potential from its current field may be neglected (see U. S. Patent 1719786, by C. Schlumberger). Measurements were first made in barren ground at station C of Figure 1. Two directions were followed, south and north (see fig. 9), which both gave the same general form of curve with ground water at a depth of about 50 feet. This water level agreed with the level of a small lake about a quarter of a mile away. The resistivity at the surface was very high, often over 1,000,000 ohms. At a depth of about 400 feet the resistivity was still high - 500,000 ohms. Another set of similar measurements was made at station B of Figure 1, a point 100 feet distant from a position directly over the ore body. At shallow depths the curves of Figure 10, show only slight variations from those of Figure 9. The general tendency of the resistivity at greater depths is to decrease continually, in one instance dropping to below 100,000 ohms, indicating a different structure at B than at C. It does not differentiate between conditions in the cardinal directions--that is, toward the south, barren ground; toward the north, across the ore body; and toward the east and west, parallel to the ore body.

The curves shown in Figure 11 were derived by using a modified method of the Gish-Rooney system, described in U. S. Bureau of Mines Information Circular 6171, in which the variation of resistance was also measured in the cardinal directions, using the middle point between the current stakes as the basis of directional resistivity and measuring the resistivity to the north and south from station B of Figure 1. The effect of the ore body is clearly shown by these curves. At a depth of 140 feet there is a divergence between the

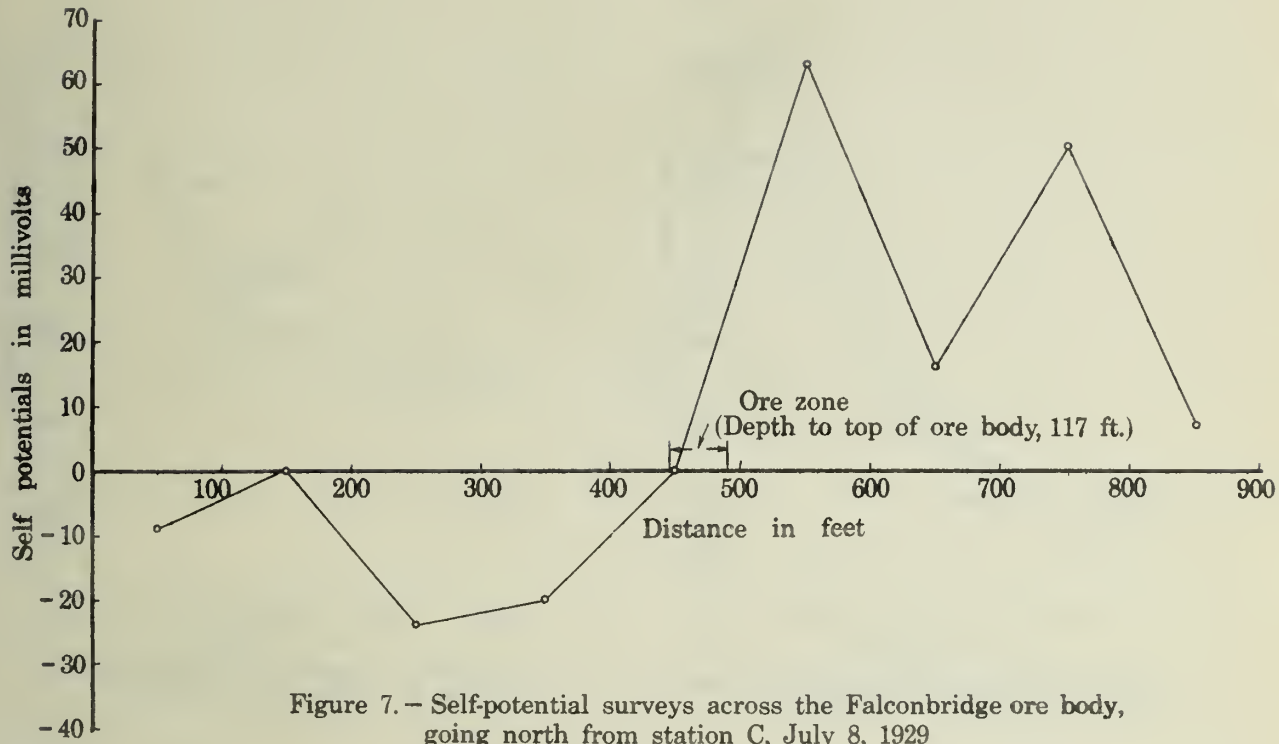


Figure 7. – Self-potential surveys across the Falconbridge ore body, going north from station C, July 8, 1929

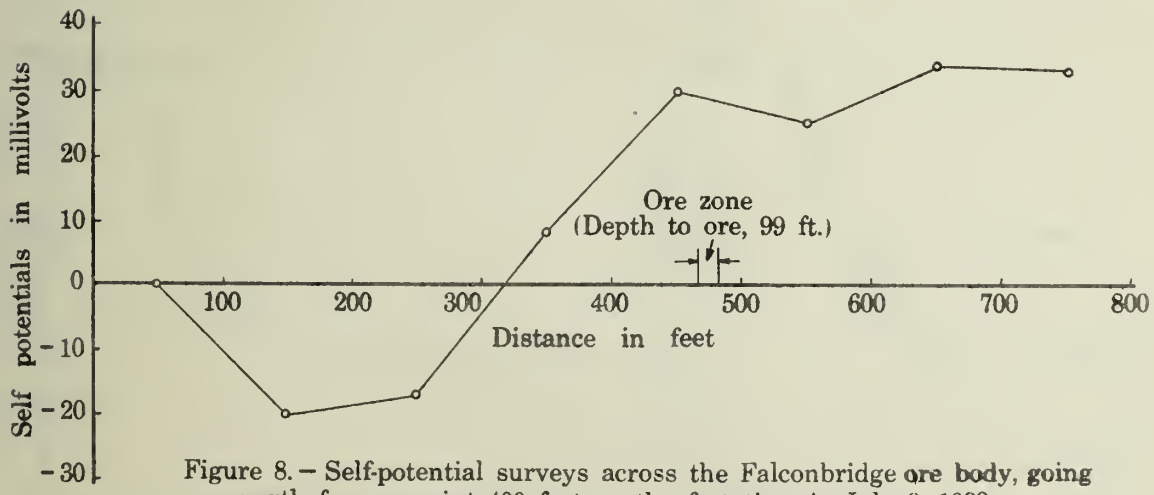


Figure 8. – Self-potential surveys across the Falconbridge ore body, going north from a point 400 feet south of station A, July 8, 1929

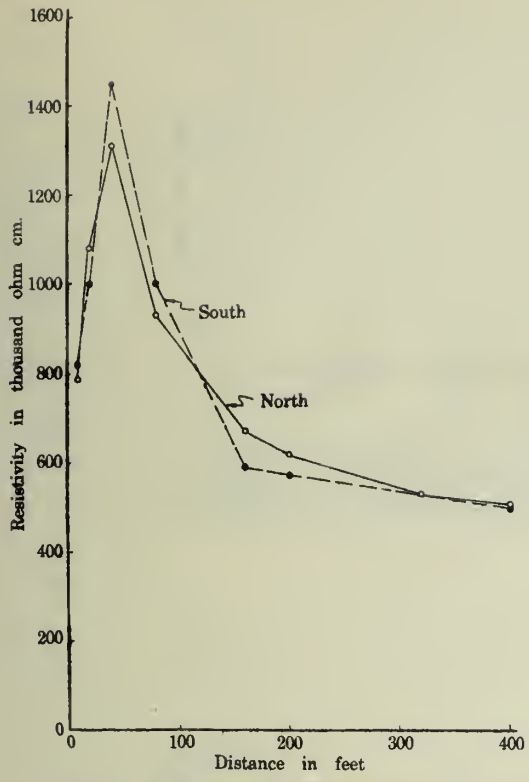


Figure 9. – Earth-resistivity measurements at Falconbridge. Single electrode probe, station C, July 8, 1929

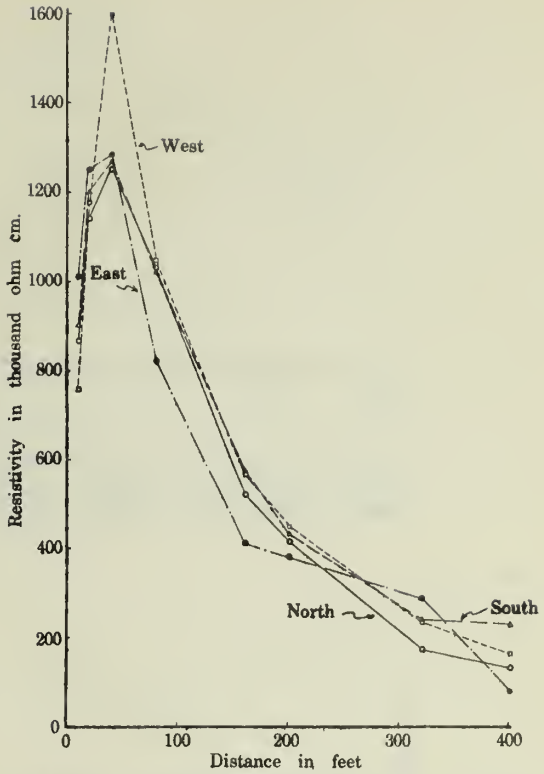


Figure 10. – Earth-resistivity measurements at Falconbridge. Single electrode probe, station B, July 8, 1929

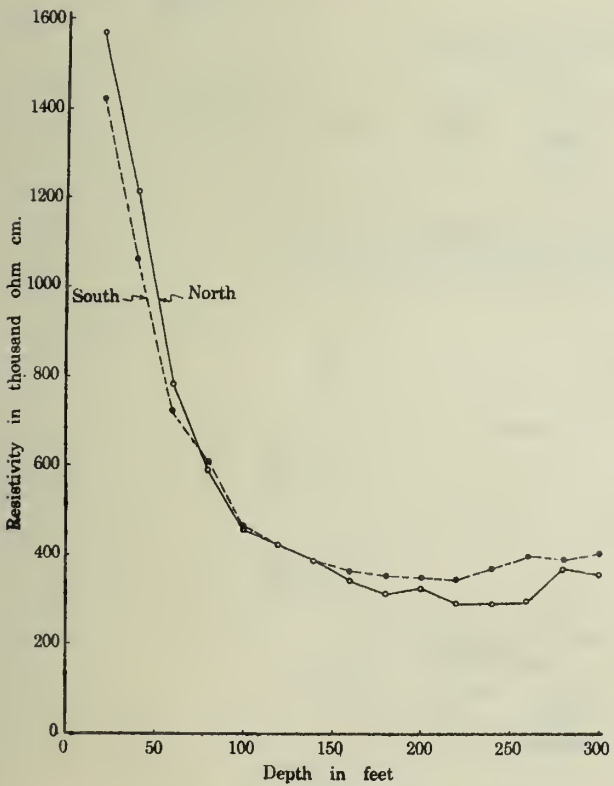


Figure 11. – Earth-resistivity measurements. Modified Rooney-Gish method, station B, July 11, 1929

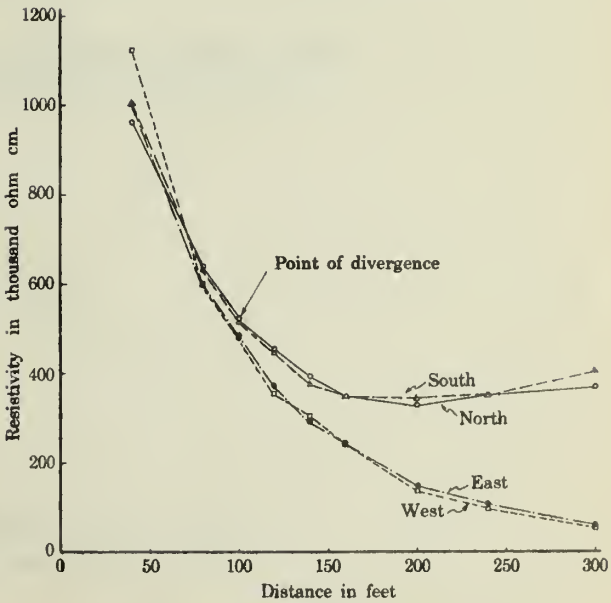


Figure 12. – Earth-resistivity measurements. Modified Rooney-Gish method, station D, July 11-12, 1929

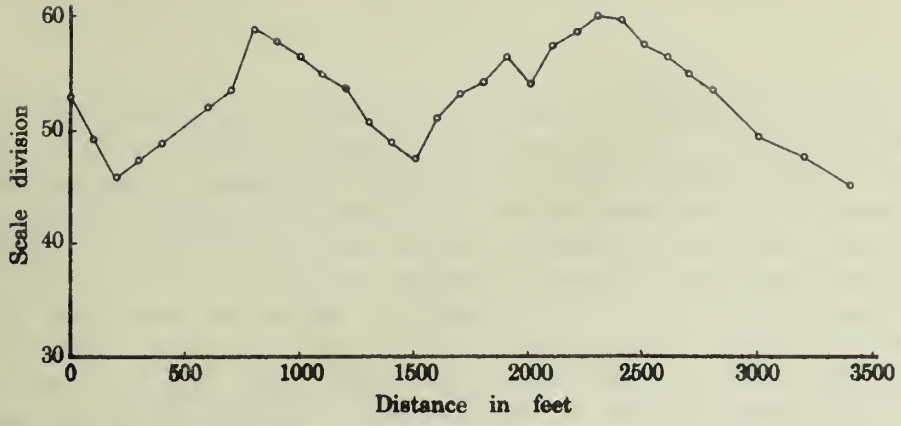


Figure 13. – Magnetic traverses across faulted zone in the Errington district. Askania vertical magnetometer measurements, starting 70 feet beyond wooden bridge over river near shaft No. 3 and going east along north ditch of road, July 16, 1929

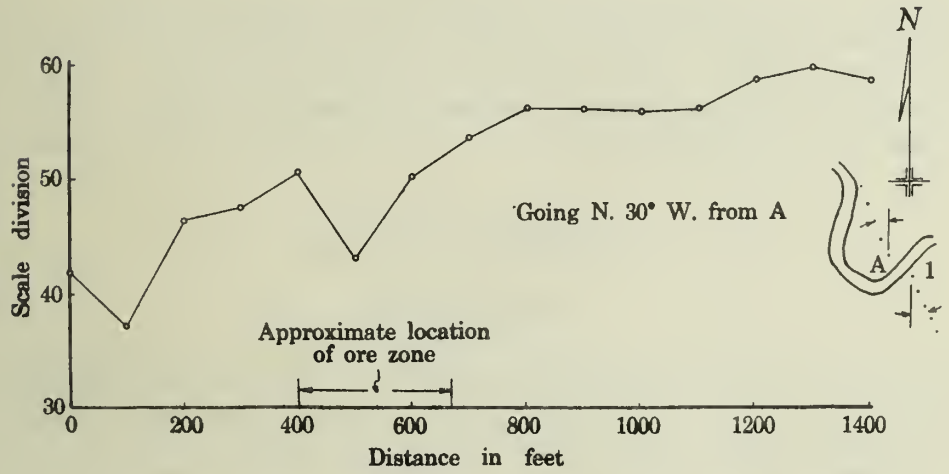


Figure 14. – Magnetic survey over ore zone with the Askania vertical magnetometer. Observations taken along line N. 30° W. from drill hole 43 near river below shaft No. 2. July 18, 1929

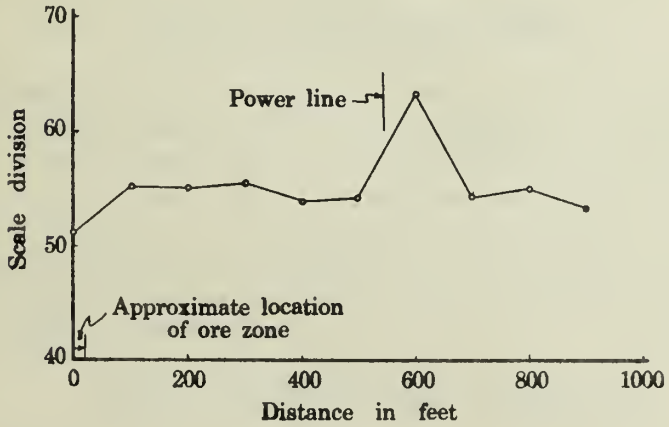


Figure 15. – Magnetic survey over ore zone with the Askania vertical magnetometer, starting about 200 feet north of station C and running north. July 18, 1929

two curves. Since the ore body is about 100 feet north and about 100 feet down, the direct distance to it is $\sqrt{(100^2 + 100^2)} = 141.4$ feet from station B. The divergence of the curves at this depth may be a coincidence and deserves further investigation. The resistivity toward the north was consistently lower after the ore body was in the measured region of potential stakes. The curves of Figure 12 were obtained by measurements made at station D, located directly over the ore body, with resistivity observations toward the north and south. This direction was transverse to the ore body, and its effect was necessarily small (See U. S. Bureau of Mines Tech. Paper 440, fig. 5.) Comparing the resistivity, measured to the east and west, with the ore body in the line of measurement to resistivity measured north and south there is a marked deviation below a depth of 100 feet. This is also the vertical distance below the surface at which the ore body occurs and further substantiates the value of exploration by resistivity methods. The extent of this ore body in depth could not be determined with the power facilities available.

As the results obtained from the three independent methods of prospecting discussed above check, there is little doubt concerning the existence of a mineralized zone as indicated in Figure 1, at about 100 feet below the surface. The decided self-potential effect is important, as it indicates the presence of sulphide ores. Furthermore, because the magnetic reaction was not too strong, the occurrence of sulphides other than iron pyrite is probable.

THE ERRINGTON DISTRICT

The Ore Deposit

The second place selected for geophysical investigation was the Errington property in the central part of the Sudbury Basin. The mineralization consists of pyrite, sphalerite, and galena in a faulted zone between a slate formation to the north and a volcanic tuff to the south. The overburden consists of sand and clay. There is a large quantity of graphitic slate and tuff in the region, and the sulphides are disseminated in a quartz and calcite gangue. A typical drill-hole record shows surface material, 43 feet; tuff, 43 to 60 feet; vein material, 60 to 166 feet; and tuff, 166 to 192 feet.

Results of Magnetic Measurements

Unlike the Falconbridge area where the vertical magnetic intensity was quite uniform this district shows pronounced variations in magnetism which can be ascribed to the presence of pyrite, of which there is much evidence in mining operations. Figure 13 shows a good example of a typical curve of a traverse over the faulted zone. In this traverse there were variations of about 15 scale divisions, equivalent to 544 gammas. Figures 14 and 15 show no special indication of increased magnetic intensity over the ore zone; on the contrary, it was below normal, but not sufficiently below normal to lead to any conclusions. Magnetometer measurements therefore have no particular value in this district.

Results of Self-Potential Measurements

Observations made along a traverse with a potentiometer and porous pots are shown

in Figure 16. The observation line runs 70 feet east of the wooden bridge shown in Figure 17, near shaft No. 3, and east along the north side of the road. With the potential gradient measured every 100 feet there is seldom a variation of over 10 millivolts plus or minus. Several factors in this region are detrimental to the proper functioning of this method. In the first place, the conductivity of the ground is very good, since its specific resistivity is under 8,000 ohms. This tends to make a good closed circuit over the oxidizing ore body so that only weak effects would extend to the surface. To some extent the bed of clay associated with the overburden prevents easy percolation of the surface waters to the ore zone and thereby reduces the chemical action. The potentials shown from B to C in Figure 16 may be considered as those of normal barren ground in this district.

A set of observations was made at another location about 3 miles away, over an outcrop plotted in Figure 18. Here very high values of self-potential were observed (over 750 millivolts in 100-foot distances) showing the self-potential method under the most favorable circumstances. Figure 19 shows another set of potential observations taken not many hundred feet away. Here the outcrop is no longer at the surface. The short-circuiting effect of the overburden limited the surface potential to a maximum of 30 millivolts. The factor which is of great importance and which indicates the presence of the ore body is the long interval of 450 feet in which there was no reversal of potential. This locates the ore zone approximately as shown in Figure 19.

Results of Resistivity Measurement

The region in which resistivity measurements were made was near No. 3 shaft, as shown in Figure 16. The ore zone extends about 200 feet north of point A and 60 to 166 feet down. Figure 20 gives the resistivity measurements made at A. The values of specific resistivities are very low, in the neighborhood of 10,000 ohms. The measurements, confirming data obtained by diamond drilling, indicate that the structure is unsymmetrical, with much higher resistivity toward the south. Further tests at station B also confirm theories regarding a complicated ground structure difficult to analyze, and indicate that further measurements in a northerly direction would be of interest. Station C, with resistivities measured toward the north and south, shows higher conductivities to the north, which is in agreement with the general direction of mineralization (see fig. 21). At D (fig. 22) the south shows the best conductivity, indicating that the conducting zone lies between D and C. Observations at E (fig. 23) indicate a conducting zone both north and south, a finding confirmed by station F in Figure 24. Another station at G, shown in Figure 25, on a line running about two points northwest, also confirms greater conductivity to the north.

Undoubtedly most of this conductivity is due to graphitic slate and tuff, the graphite of which makes them conductors. Consequently it is possible to locate ore in this formation only when it occurs in conjunction with the graphite. That the results were as good as indicated was rather unexpected.

This district, considered as a whole, is unsuited to geophysical observations by magnetometer methods as practiced to-day, and resistivity methods offer only a poor guide to the discovery of shallow deposits having a self-potential associated with them.

The results obtained with these three different methods of geophysical prospecting,

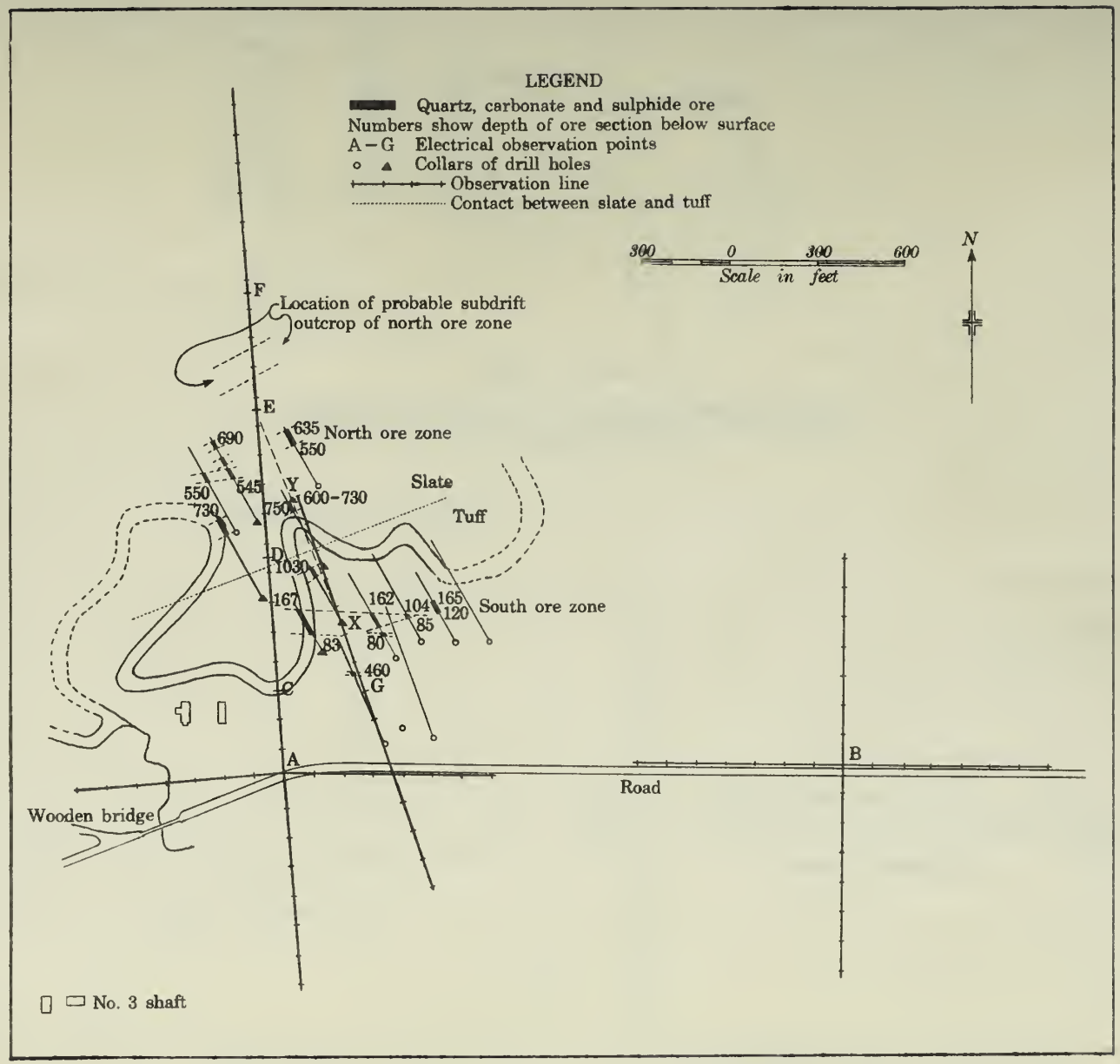


Figure 16. – Topographical chart of the Errington district

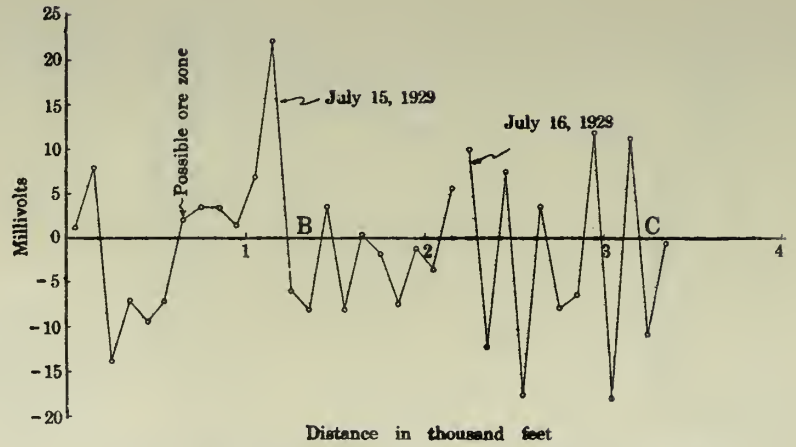


Figure 17. – Self-potential survey in the Errington district, starting 70 feet east of wooden bridge over river near shaft No. 3 and running east along north side of road. July 15–16, 1929

NOTE: Run over exposed sulphides

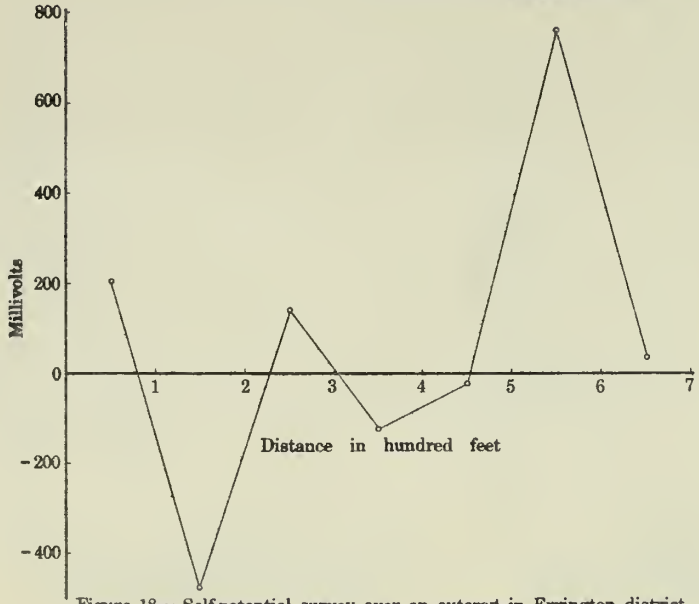


Figure 18. – Self-potential survey over an outcrop in Errington district. July 20, 1929

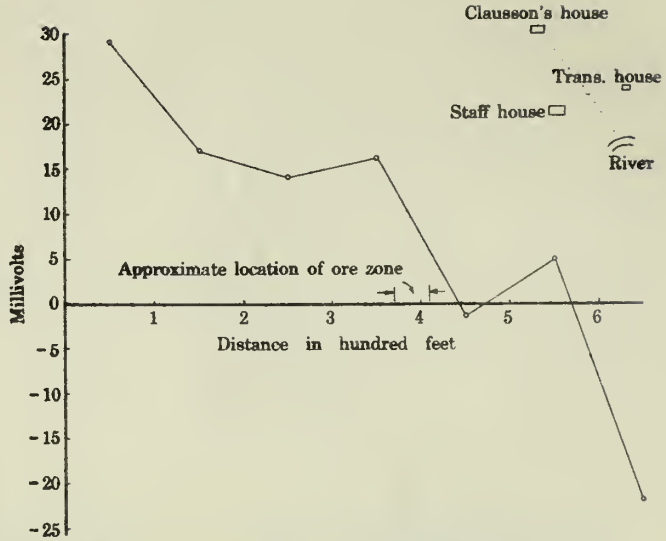


Figure 19. – Self-potential survey near outcrop, starting at river and running over to Clausson's house, July 24, 1929

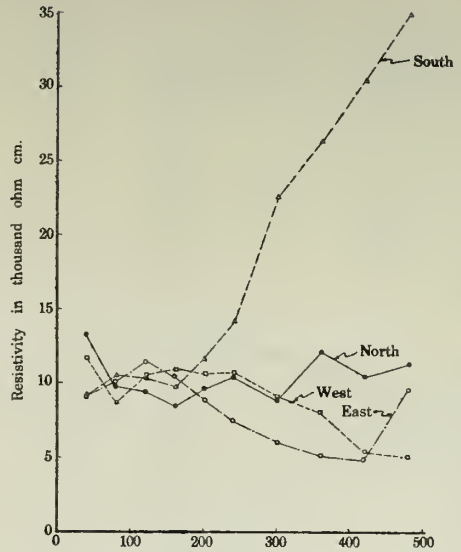


Figure 20. – Earth-resistivity measurements in the Errington district. Modified Rooney-Gish method, station A. July 15-16, 1929



Figure 21. – Earth-resistivity measurements in the Errington district. Modified Rooney-Gish method, station C, July 18, 1929

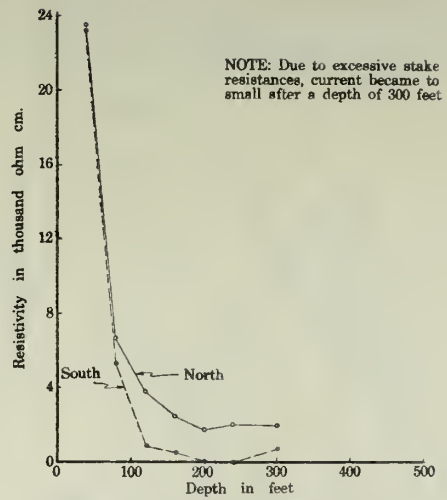


Figure 22. – Earth-resistivity measurements in the Errington district. Modified Rooney-Gish method, station D, July 18, 1929

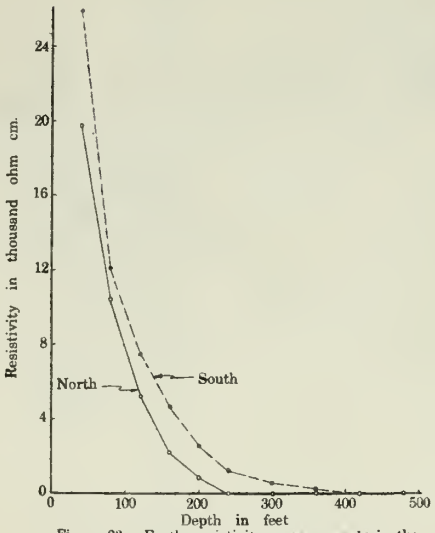


Figure 23. – Earth-resistivity measurements in the Errington district. Modified Rooney-Gish method, station E, July 19, 1929

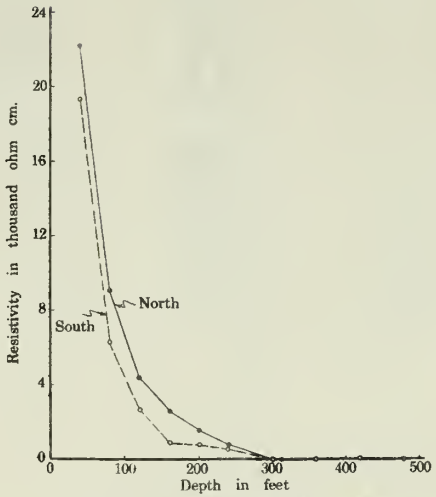


Figure 24. – Earth-resistivity measurements in the Errington district. Modified Rooney-Gish method, station F, July 19, 1929

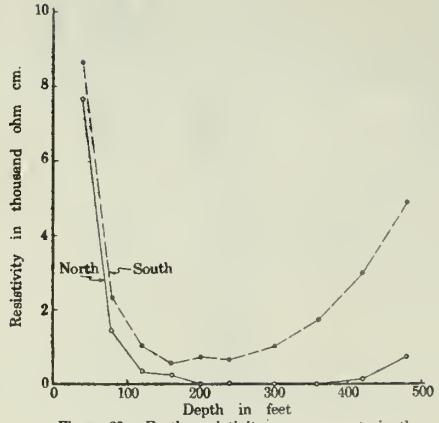


Figure 25. – Earth-resistivity measurements in the Errington district. Modified Rooney-Gish method, station G, July 20, 1929

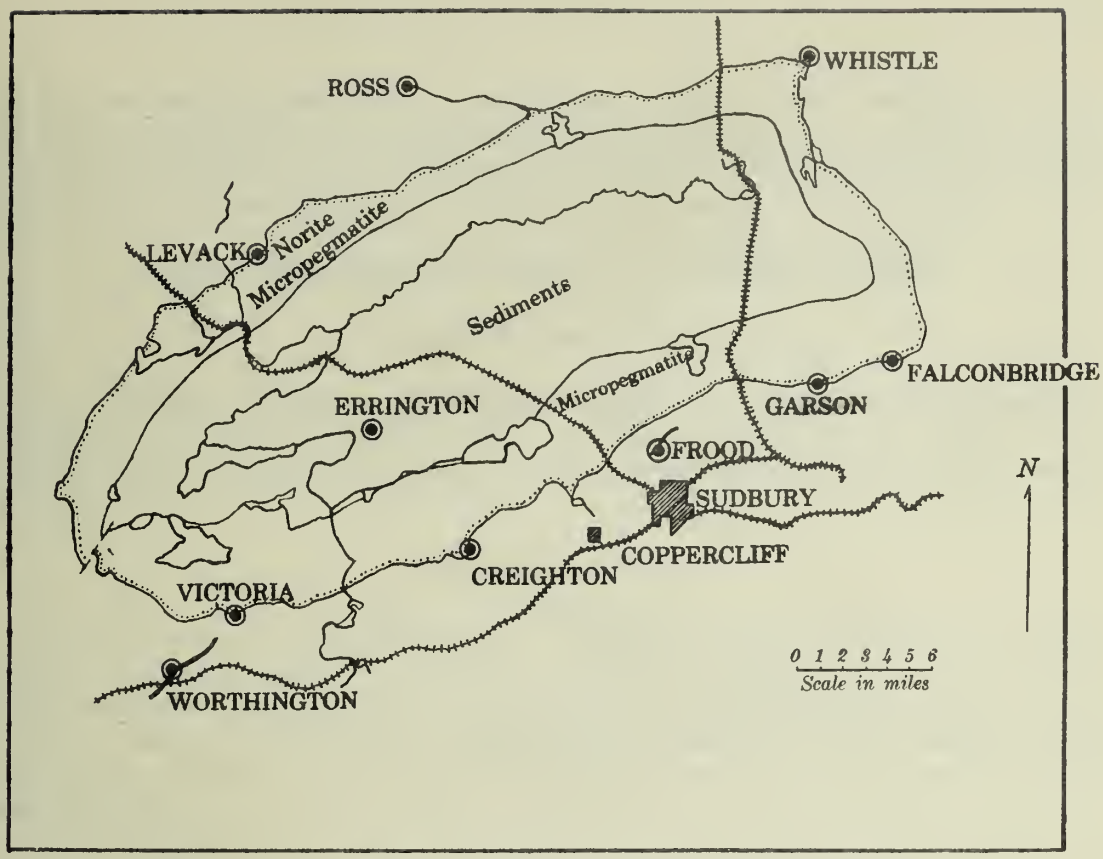


Figure 26. – Topographic map of Sudbury Basin, showing outline of micropegmatite norite sheet and overlying central sedimentary area

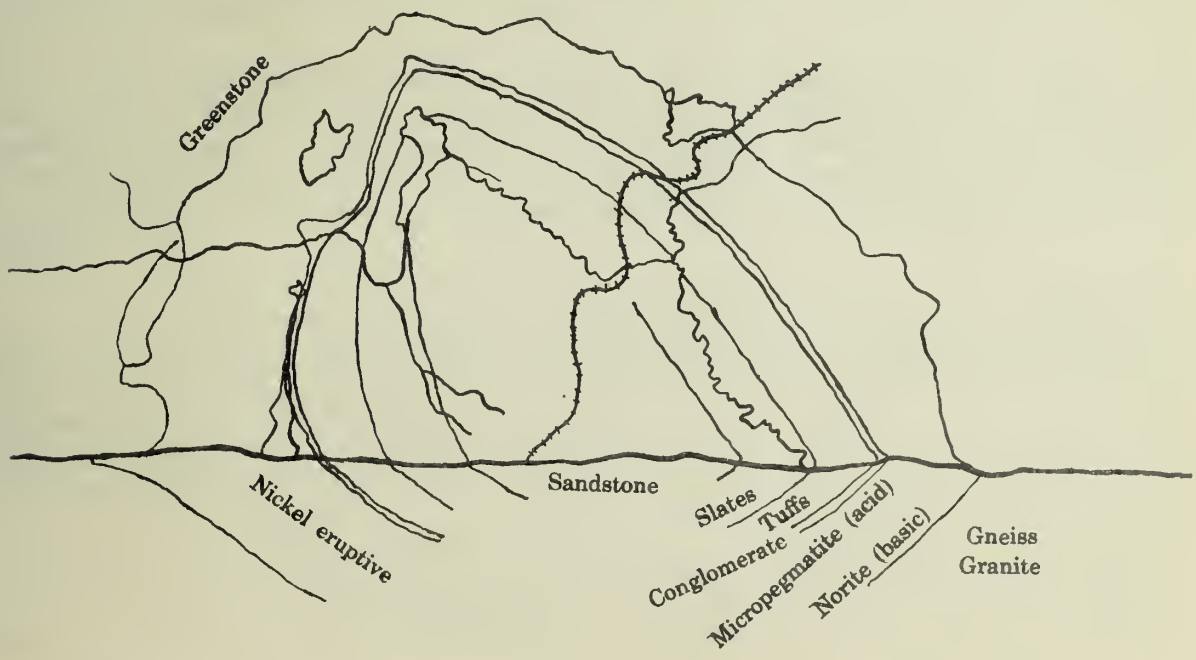


Figure 27. – Geological cross section through Sudbury nickel region looking southwest



indicate that the Falconbridge area is much better suited to their application than the Errington area, especially since the conductivity of the ores in the latter is very low. A mine operator should know how far his particular district will lend itself to geophysical prospecting methods and the difficulties encountered in applying these methods.

DESCRIPTION OF MINES AND DISCUSSION OF GEOLOGICAL FEATURES

By J. B. Mawdsley[3]

As the general geology of the Sudbury Basin has attracted world-wide attention and as the literature concerning it is so replete, this paper is limited to a description of the mines where geophysical studies were made and to a discussion of their salient geological features.

FALCONBRIDGE MINES

The Falconbridge Nickel Mines, Ltd., control a tract of territory in Garson and Falconbridge Townships along the norite contact near the sharp-pointed southeast corner of the boat-shaped Sudbury basin (fig. 26). This contact is sinuous, but, in the south parts of lots 10, 11, and 12, Concession IV, Falconbridge Township, it has a general east-west trend. Along this contact a nickel-copper sulphide ore body 7,500 feet long has been traced by diamond drilling.

The 1,000-foot shaft of the Falconbridge mine is in the southwest corner of lot 12, about 150 feet south of the ore body. About 3 miles southwesterly from this shaft in Range III, near the south boundaries of lots 4, and 5, Garson Township, is the Garson mine, which has been an important producer for many years.

During 1916 and 1917 the E. J. Longyear Co. of Minneapolis, Minn., former owners of this property, carried out an extensive prospecting campaign by diamond drilling and delineated the ore zone. The large amount of information gathered by this company regarding the relationships of the ore body to its surrounding rocks has been ably summarized by Roberts and Longyear.[4] The writer has drawn freely from this article in the following description.

It is interesting to note that the magnetic property of the nickel-copper Sudbury ore due to its pyrrhotite content has long been known and used in prospecting for these ore

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3 - Geologist, Canadian Geological Survey.

4 - Roberts, H. M., and Longyear, R. D., Genesis of the Sudbury Nickel-Copper Ores. Trans. Am. Inst. Min. Eng., vol. 59, 1918, pp. 27-56.

bodies. In 1856[5] a surveyor noted the magnetic effect of pyrrhotite in the vicinity of the present Creighton mine. Since 1893, when a railway cut on the main line of the Canadian Pacific Railway northwest of Sudbury, intersected the outcrop of what later was developed as the Murray mine, magnetic methods have been used in prospecting for nickel-copper deposits in this district. The instrument generally employed was the magnetometer. Coleman cites one difficulty in interpretating correctly the results of these magnetic surveys, namely, the variable amount of magnetism in the pyrrhotite of this district. These surveys have nevertheless proved of considerable value and have been used systematically by at least two companies in the region.[6] In the vicinity of the Falconbridge property an extensive magnetic survey was made under the direction of T. A. Edison. No deposit was discovered as a result of this survey, for the test pit started on what is now the Falconbridge property, never reached bedrock, owing to quicksand. This test pit, which still can be seen, was sunk about 200 feet north of the present Falconbridge shaft, directly over one of the widest sections of the ore body on this property.

Topography and Superficial Deposits

In the vicinity of the shaft and over a section of territory extending about 4 miles to the east the surface relief is well defined by a remarkable series of kettle holes. These kettle holes are of various sizes, between 350 and 1,300 feet in diameter, and have steeply sloping regular sides and depths of as much as 120 feet. They are separated from each other by very narrow ridges. These regular inclosed basins contain no ponds or muskegs, as their sides and bottoms are of gravel and coarse sand which allows the moisture to seep through to lower drainage channels some distance away.

East of this area, which resembles a series of volcanic craters set closely together, is a comparatively flat plain with a gentle slope to the southeast. The elevation of this plain is only slightly lower than the top of the ridges separating the kettle holes, which have an elevation of about 970 to 1,010 feet above sea level. There are no outcrops of rock near the shaft or for a mile east of it on the level plain just mentioned.

Drilling has indicated that the drift covering the underlying rock is 50 to 250 feet deep. In the area studied electrically, which is on a section of the flat plain just described, the thickness of the drift was found by drilling to be 100 feet to 156 feet.

The material forming this drift is coarse gravel, sand, and occasionally large bowlders. That water action had something to do with its deposition can be seen by a study of excavations in this material, which often display fine examples of bedding.

There is little doubt that this drift and topography were formed in the retreat of the Pleistocene ice sheet which once covered this region, together with nearly all eastern Canada and adjacent sections of the United States.

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5 - Malcolm, Wyatt, and Robinson, A.H.A., Canada: Geology, Mines and Metallurgical Industries. Dept. of Mines, Ottawa, 1927.

6 - Coleman, A.P., The Nickel Industry. Mines Branch, Dept. of Mines, Ottawa, 1913 p. 126.

Geology

Drilling at intervals of 200 feet in an east and west direction near lot line 10/11 and concession line III/IV a little over half a mile east of the Falconbridge shaft shows that the sulphide body over which the electrical work was done lies along the east-west contact between the norite to the north and the quartzite-graywacke of the Sudbury series to the south. Drill holes show that the ore body dips 80° to the north.

Roberts and Longyear[7] state that the quartizite-graywacke formation is composed of about equal portions of quartz grains and altered ferromagnesian minerals. In some places the quartz is in excess and the rock grades toward a true quartzite, while in others the ferromagnesian minerals predominate, giving a more typical graywacke. Some phases of the formation have a marked schistosity approximately parallel to the bedding, which is stated by them as north 80° east and having a vertical dip.

A highly chloritized greenstone is at times found associated with the quartzite-graywacke formation. Some of it appears in the vicinity of the shaft but is not mapped as occurring near the area electrically surveyed. Quantities of this rock may nevertheless occur south of the ore zone in areas not prospected by drilling.

The norite, which is younger than and intrudes into the two formations just mentioned, occurs on the Falconbridge property. It is a rock of medium grain and granitic texture[8], and is somewhat more acid in composition than the norite usually found in the Sudbury district. The feldspar is andesine (An_{40}) to oligoclase (An_{25}). The pyroxenes have been replaced almost completely by hornblende, uralite, chlorite, and serpentine. A small amount of olivine is sometimes present. Quartz usually forms an appreciable proportion of the rock and occurs as interstitial grains and as fine aggregates due to crushing by dynamic forces. The quartz has the typical opalescent blue color characteristic of the Sudbury norite. Considerable brown biotite is always present.

Blebs of pyrrhotite and chalcopyrite are visible in the ore, but traces of these minerals are seldom found more than a few hundred feet from the ore. The writer noted that near the 1,000-foot level of the Falconbridge mine the contact between the sulphide ore and norite was sharp and that the adjacent norite carried not more than 7 per cent of the sulphides. The norite carries minor amounts of apatite, magnetite, and accessory constituents.

Typical of the rest of the ore zone on the Falconbridge, the ore body in the vicinity of the locality surveyed electrically is, as previously mentioned, dikelike in form and dips about 80° north. As indicated by diamond drilling it is 7 to 45 feet wide along this part of the zone a short distance below the rock surface. From the evidence already obtained in the 1,000-foot shaft and from general considerations it is to be expected that this continuous body of sulphides not only will be long horizontally but probably also will extend to considerable depths. It is also to be expected that the ore body will probably vary in width along its downward extension.

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7 - Roberts, H. M., and Longyear, R D., Work cited, p. 33.

8 - Roberts, H. M., and Longyear, R. D., p. 34 of work cited.

The sulphide minerals comprising the ore are, in order of abundance, pyrrhotite ($Fe_{11}S_{12}$), pentlandite ($FeNiS_2$), and chalcopyrite ($CuFeS_2$). The proportion of the various sulphides to each other varies, but the average proportion is probably about 80 per cent pyrrhotite, 14 per cent pentlandite, and 6 per cent chalcopyrite; a little magnetite and pyrite are present. It has been determined that the proportion of sulphides between the foot and hanging wall of the ore zone ranges from 25 to 56 per cent.[9] The balance of the ore zone is formed of silicates, usually in the form of fragments of any of the country rocks present in the area--norite, quartzite, greenstone, and quartz. According to Roberts and Longyear, these fragments may be rounded, subangular, or angular and generally suggest replacement by the sulphides.

ERRINGTON MINE

The Readwell Yukon Co., Ltd., is carrying out extensive mining operations at its Errington mine in the central part of the Sudbury Basin (fig. 26). This property consists of nearly 7 square miles, and its center lies about 13 miles west and 3 miles north of the town of Sudbury.

Topography and Superficial Deposits

Trending east-northeast and west-southwest and forming the south half of the property are low irregular wooded hills of volcanic ash or tuffs. The three shafts are situated on the north flank of this rocky area. North of these hills is a flat plain, mostly cleared, and forming the southern margin of the farming country that occupies the central part of the basin. Traversing this plain a few hundred feet north of shafts Nos. 2 and 3 and flowing southwestward between shafts Nos. 1 and 2 is a meandering stream 20 to 40 feet wide sunk about 20 feet below the level of the plain, with small rapids just east of shaft No. 1. Near shafts Nos. 2 and 3 the ore zone lies below the valley of the stream.

The surface material of the part of the plain north of the shafts is composed of extremely fine sandy silt containing considerable clay. This material was deposited in a postglacial lake. When saturated with water it is a veritable quicksand. Although the surface appears dry and baked, the watertable must be close below it.

Over the ore zone south of shaft No. 1 the drift is shallow and is only a few feet thick. Northwest of shaft No. 2 its depth, as indicated by drilling, is 50 to 118 feet about 300 feet south of the ore zone. It is probably about 100 feet thick over the ore zone. Northeast of shaft No. 3, where most of the electrical work was done, the depth of this drift is 45 to 130 feet.

Geology

No detailed description of the geology near the mine has been published, and the

9 - Roberts, H. M., and Longyear, L. D., work cited, see fig. 6

writer is indebted to the officials of the company for much of the following information and for elaborate surface and underground maps of the property, which lies wholly in the sediments within the igneous ring forming the Sudbury Basin. As previously mentioned, the inner margin of this intrusive is micropegmatite. The three shafts lie about 2 miles north of this inner contact.

About 800 feet north of shaft No. 1 and about 1,300 feet north of shaft No. 2 is believed to be the axis of a fault zone having a strike of 71°. This fault is apparently the east-northeast extension of what is termed the Cameron Creek fault, which intersects the southeastern apex of the igneous margin of the basin some 13 miles to the west-southwest.

The contact between the black Onwatin slates and the fine, dark Onaping tuffs appears to be approximately along the fault zone. Both are members of the Whitewater series of late pre-Cambrian age and form the sedimentary series underlying the central part of the Sudbury Basin.

The Onwatin slates exposed on the property are fine black slates with a fairly good cleavage which is usually rather flat and is believed to parallel the bedding closely. These slates contain notable quantities of graphite, a sample from this property showing 2.92 per cent.[10] Slips in these slates are common, are coated with graphite, and contain a much higher percentage of this mineral than the above sample (see fig. 27).

The Onaping tuffs seen on the property are often fine-grained, and slaty-looking varieties can not be distinguished from the Onwatin slates. Coarser tuffs have distinguishable fragments of volcanic rocks in a fine-grained, dark matrix. Generally speaking, the graphite content of these rocks is not as high as that of the slates. When fine-grained, the tuffs are distinguishable by their fine spotted appearance, which is due to the fragments they contain.

Near the fault zone what are believed to be tuffs often contain white specks about a millimeter in diameter. These particles are calcite or siderite, and their presence is probably due to the action of the mineralizing solutions that invaded the fault zone.

The bedding of these rocks is hard to distinguish. North of shaft No. 3 the rocks are sheared, with a strike parallel to the fault zone and a dip of about 60° east-southeast.

The ore zone has been traced throughout the length of the property and lies along the fault zone already mentioned. The zone has been found to be sinuous and is in the form of irregular bands of varying widths and with poorly defined ends. The structure of this zone is complicated, and it is difficult to find and develop the ore bodies.

The sulphides, chiefly pyrite, sphalerite, and galena, with occasionally a little pyrrhotite, occur both disseminated and in solid bodies of various irregular shapes in a quartz and calcite gangue. The gangue minerals form a considerable portion of this zone, although shreds and fragments of the country rock are common, and graphite is usually present. The proportion of the sulphides to each other and to the inclosing gangue vary considerably in the different ore bodies. The micrographic relationships of the various minerals were not studied.

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10 - Coleman, A. P., Moore, E. S., and Walker, T. L., Sudbury Nickel Intrusive. Univ. of Toronto Studies, Geol. Series No. 28, 1929, p. 39.

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FEBRUARY, 1930

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MILLING PRACTICE AT THE ALASKA JUNEAU CONCENTRATOR

BY

P. R. BRADLEY

February, 1930.

To the reader:

This is one of a new series of papers that will present the details of milling practice and itemized costs at the principal ore-dressing plants in the United States. Officials and mill superintendents of the mining companies will prepare these papers, in cooperation with the mining division of the Bureau of Mines, and in accordance with a bureau outline designed to obtain uniform presentation.

The facts thus assembled have immediate value to the Bureau's research engineers working on ore-dressing problems and will be the basis for summarized reports by the bureau on methods and costs of separate steps in milling, such as crushing, grinding, screening, conveying, and concentrating by gravity, flotation, and leaching. Publication in mimeographed form allows these papers to be issued within a short time after receipt, and it is hoped they will be welcomed as cordially by the mining public as was the series on mining methods and costs.

Here again, we are counting on the hearty assistance of the industry, and the result should be another fine example of coordinated effort between industry and government, for the good of all concerned.

Scott Turner

SCOTT TURNER,
Director.

I.C.6236.
February, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE--BUREAU OF MINES

MILLING PRACTICE AT THE ALASKA JUNEAU CONCENTRATOR[1]

By P. R. Bradley[2]

This paper, describing the milling practices at the Alaska Juneau concentrator, Juneau, Alaska, is one of a series on milling practices being published by the Bureau of Mines.

ACKNOWLEDGMENT

A large part of the paper was prepared by Walter P. Scott, mill superintendent, Alaska Juneau Gold Mining Co., Juneau, Alaska.

GENERAL DESCRIPTION

The concentrator of the Alaska Juneau Gold Mining Co. is located on the steeply sloping shore of Gastineau Channel, near Juneau, Alaska. The Alaska Juneau mine is in Gold Creek Basin, about 2 miles from the mill. The ore is mined by a caving system, dropped through ore passes, and trammed in 10-ton cars through the haulage level to the tipple at the top of the mill.

The mill is an example of an extreme type of hillside construction, the difference in elevation between the tipple and the bottom floors being 263 feet. Very little elevating of material is required. The ore is free milling and does not require very fine grinding to free the values; the flow sheet is quite simple, consisting primarily of crushing, sorting, grinding, and table concentration. Approximately 11,000 tons of run of mine material is trammed to the mill daily, and 46 to 48 per cent of this is rejected as waste by hand-sorting.

Water for milling purposes is obtained from Gold Creek and delivered through a flume and a 20-inch wooden stave pipe line to a surge tank above the level of the ball-mill floor; 8,000 to 8,500 gallons per minute is required. No recovery of water is made, as it is all required to convey the fine tailings through the tailings flume to the beach. During very cold weather, when the supply of fresh water is inadequate, salt water is pumped from Gastineau Channel. Salt water is more satisfactory than fresh water during extreme cold weather, because it does not freeze so readily.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6236."

2 - One of the consulting engineers, U. S. Bureau of Mines.

Power, supplied by three hydroelectric plants owned by the company, is generated at 2,300 volts and stepped up to 22,000 volts for transmission to the central power plant. During shortages of water additional power is purchased from the Alaska Gold Mines Co. A stand-by oil-fired steam plant at the central power plant, equipped with one 2,000-kw. and one 6,000-kw. General Electric steam turbine, is used during interruptions on transmission lines and other emergencies. The average mill load is about 3,000 kw., and the total load for the entire operation 5,000 kw. All motors in the mill over 15 hp. are 2,300 volt, 3 phase, 60 cycle; all 15 hp. and under are 220 volts.

Gold, the principal economic metal, is mostly native and usually associated with small quantities of galena and sphalerite. The principal sulphide, by volume, is pyrrhotite with some pyrite. The gold is quite coarse, an occasional particle weighing over 1 gram.

The ore occurs in irregular stringers and gash veins of quartz distributed through black slate and brown metagabbro dikes. Coarse particles of native gold occur most commonly at the contact of the quartz and the gabbro, although coarse gold is found in the quartz. The metagabbro, the principal gangue material, is extremely hard and tough and very hard to crush. It tends to break into slabs, adding to the difficulty of handling. The ore as delivered to the mill varies from slime to slabs that will pass a 27-inch grizzly in the mine bulldozing chambers and has a moisture content varying from a few per cent to saturation. Most of the ore cars arrive at the mill partly full of water, and this water causes excessive wear on feeders and all coarse-crushing equipment.

The present plant was put into operation April 1, 1917, and was designed to treat 8,000 tons of run-of-mine ore per 24 hours; but owing to the failure of some experimental features the expected capacity could not be realized, and the cost of milling was several times greater than anticipated. Hand-sorting was introduced, and a vigorous campaign of alteration began which, together with sorting, has increased the capacity of the plant to such an extent that an average of approximately 13,000 tons per day was handled during part of 1927, 57 per cent of this material being rejected as waste by hand-sorting. The present fine-grinding capacity of the mill is about 6,000 tons per 24 hours. The following table gives the progress made in increase of capacity and reduction of costs since the plant went into operation.

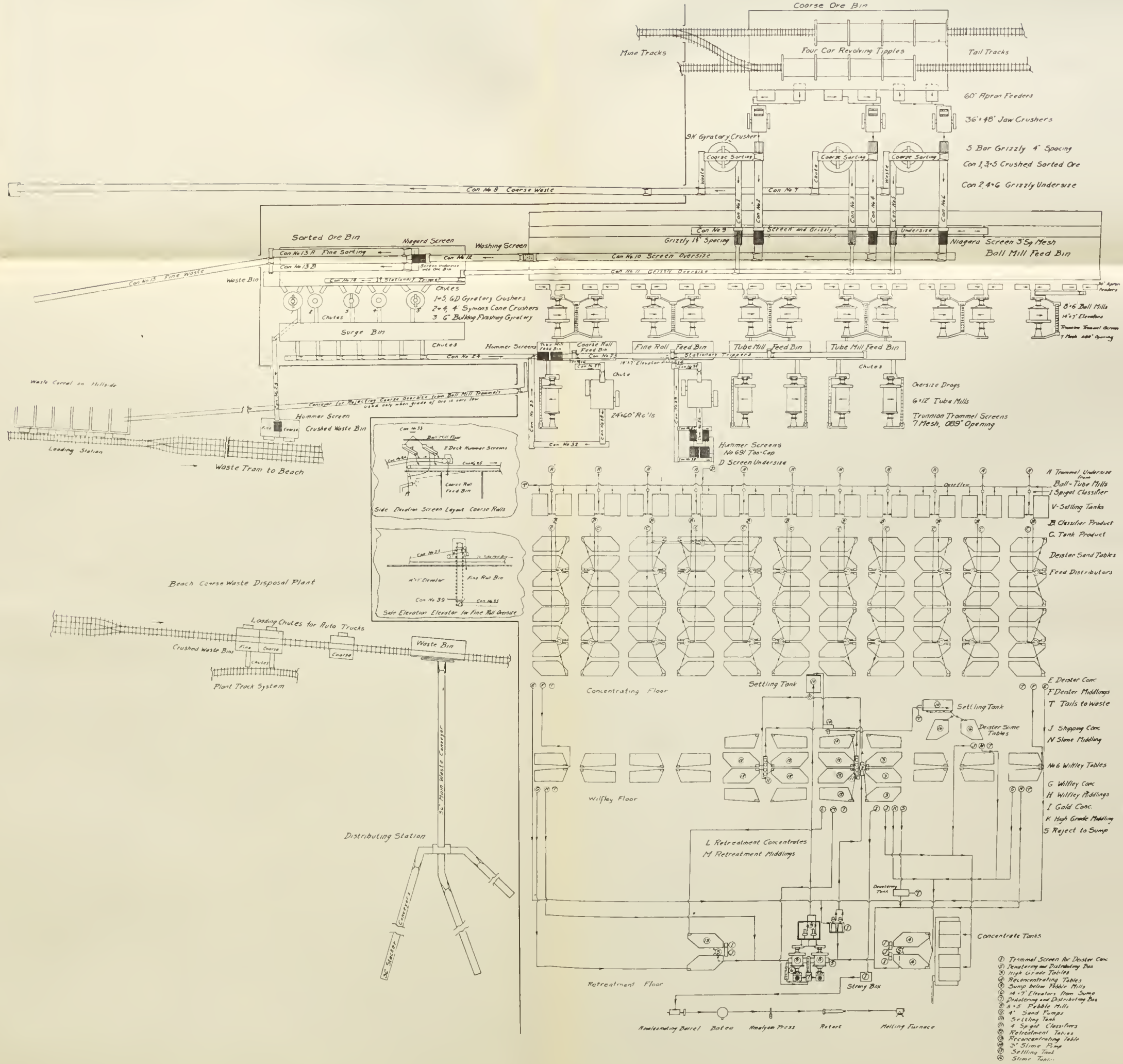


Figure 1. - Flow sheet, Alaska Juneau Mill

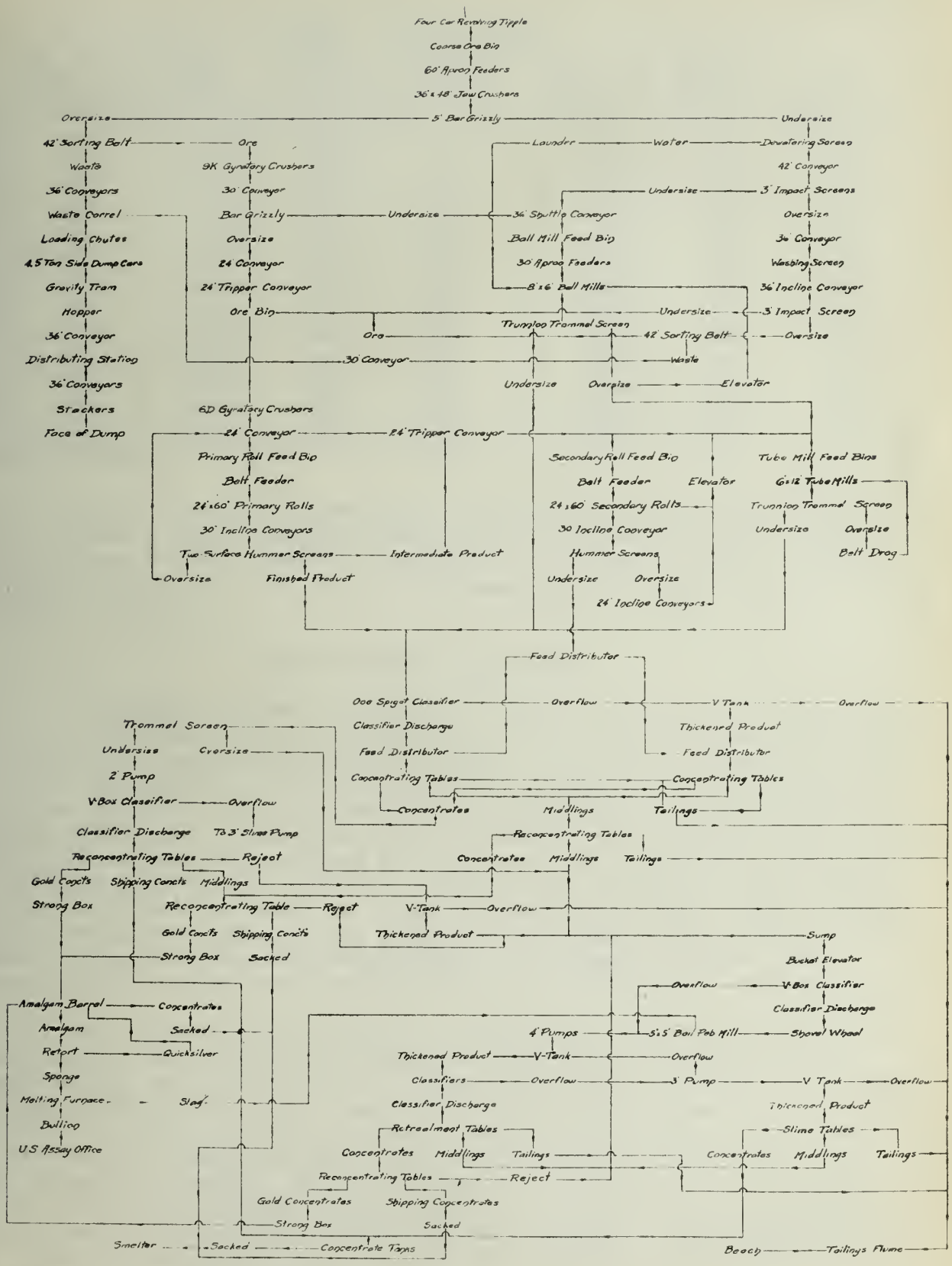


Figure 2.- Flow sheet, Alaska Juneau Mill

Year	No. of days in operation	Tons mines per day	Tons fine milled per day	Tons coarse waste rejected per day	Cost of milling	Gold assay value	Gold recovered
1917	274	2472	2472	-	$0.7171	$0.86	$0.64
1918	363	1631	1582	49	.6762	.92	.73
1919	363	1909	1698	211	.5608	.99	.72
1920	364	2590	1751	839	.4463	1.08	.78
1921	363	4445	2491	1954	.3554	.86	.60
1922	361	6400	3070	3330	.2613	.81	.56
1923	361	6860	3144	3716	.2662	.81	.58
1924	362	8476	3778	4698	.2512	.87	.62
1925	362	9618	4248	5370	.2504	.85	.58
1926	362	10579	4557	6022	.2154	.75	.50
1927	362	11790	5083	6707	.2257	.77	.55
1928	363	10243	4946	5297	.2285	1.11	.84
1[1]929	271	10718	5640	5078	.2363	1.10	.86

1 First 9 months.

The percentage of coarse waste rejected since 1927 has decreased, due to a change in the character of the ore delivered to the mill. The flow sheets, Figures 1 and 2, show the arrangement of equipment in the plant and the flow of the material; the vertical arrangement of the mill is shown in end elevation, Figure 3.

COARSE CRUSHING AND SORTING

Tramming continues throughout the 24 hours of the day, but the crushing plant is operated only two 8-hour shifts, with 3 hours between shifts. One hour is taken out during each shift for lunch. Four cars are dumped at a time by the revolving tipple into the coarse-ore bin, which has a live capacity of 1,500 tons, so there is little leeway for interruptions in either tramming or crushing. The rock is drawn from the bin by six 60 inch Stephens-Adamson apron feeders, two for each of the three 36 by 48 inch type C Buchanan jaw crushers, which are set at 8 inches. Only two crushing units are operated at one time, the third being held in reserve to allow necessary repair work. There is no screening of material ahead of the crushers. Occasional large slabs block the feeders or hang up in the crushers and have to be blasted in place. The crusher discharge falls upon a grizzly 5 feet in length with 4 inch openings.

The oversize of this grizzly is washed with a spray and drawn over a 42-inch sorting belt traveling 100 feet per minute. This sorting belt is placed directly over the bowl of a 9K Gates-type gyratory crusher set at about 2 inches. Four sorters work at each belt and pull off all quartz and any rocks with adhering quartz; the black slate and barren gabbro remain on the belt, are delivered to a chute leading to a 36-inch conveyor, and go thence to a waste bin on the hillside. The quartz from the sorting belt drops directly into the bowl of the gyratory crusher, and the crushed product is conveyed to a fine grizzly 7 feet in length with 1-1/2-inch openings. The undersize of this grizzly is distributed into the ball-mill feed bin, and the oversize is conveyed to the "sorted-ore" bin.

The undersize of the 4-inch grizzly is passed through a chute equipped with cast screen-bottom liners to remove as much water as possible and then to a 42-inch conveyor which delivers it to a 3 by 6 foot Niagara vibrating screen equipped with 3-inch square-mesh cast-manganese screens. The undersize of these screens is distributed by a 36-inch shuttle conveyor into the ball-mill feed bin, which has a live capacity of about 5,000 tons. The water screened from the ore is run in launders to the feed boxes of the ball mills.

The oversize of the Niagara screens is conveyed to a stationary washing screen, then elevated by an inclined conveyor to a second Niagara screen having the same size and type of screen as the first. The undersize of this second screen drops into the sorted-ore bin, and the oversize is passed over two 42-inch sorting belts traveling 150 feet per minute and located over the top of the sorted-ore bin. Four sorters at each belt pick off the ore, dropping it directly into the sorted-ore bin, and the remaining waste is conveyed to the waste bin. The last Niagara screen removes the undersize, which passes over the first coarse screens as a result of fluctuations in the rate of feed. The amount varies from 2 tons per minute to short surges of 8 to 10 tons per minute. Niagara screens have proved very satisfactory, even though they work under such severe conditions. The cost of their upkeep and maintenance is 0.2 per ton handled.

The proportion of oversize on the 4-inch grizzlies ranges between wide limits, depending upon mine and weather conditions. During very rainy weather an excess of fine material is washed down through the open stopes. At present the average amount of oversize will run from 40 to 45 per cent of the material trammed from the mine. Ore sorted from this material will average 1,000 to 1,200 tons per day. The average feed to the ball-mill bin is 4,200 tons per day and 500 to 700 tons screened and sorted into the sorted-ore bin. The maximum tonnage that has been put through the crushing plant in any 24-hour period was 15,600 tons.

An operating crew for each shift in the crushing and sorting department comprises a foreman and 32 laborers. All laborers on crushing and sorting are Indians and Filipinos. Maintenance and repair work is handled by a separate crew.

The crushers and gyratories are all belt-driven. The original installation had one 350-hp. synchronous motor direct-connected to a line shaft, equipped with Allis Chalmers friction clutches, driving both the crusher and gyratory in each unit. One unit is still operated in this way, but the other two units have individual 150-hp. motors for each machine. Crushers are driven at 166 r.p.m. and gyratories 300 r.p.m.

DISPOSAL OF WASTE

The rejected waste is drawn from the bottom of the bin through chutes with underslung arc-type gates into 4-1/2-ton Granby-type cars. These cars are handled in two trains of nine cars each on a double-track balanced-gravity tram and deliver waste to the beach, about 1/2 mile away, where it is automatically dumped into a receiving bin. This bin feeds a 36-inch conveyor, which carries the material out over the waste dump to a distributing station. Three diverging

36-inch conveyors, terminating in stackers, lead out from this station and deliver the waste to the face of the dumps several hundred feet out into the waters of Gastineau Channel. The steeply sloping bottom of the channel has provided an excellent dumping site, and the resulting dike of waste rock has made a successful breakwater for the water front of Juneau.

All waste is trammed in two 8-hour shifts; an operating crew consists of a shift boss, hoistman, and nine men. Repair work, moving stackers, etc., are done by a separate crew.

All conveyors are belt-driven with gear and pinion. The three stacker conveyors have tandem drives a few feet ahead of the tail pulley. Each conveyor is driven by a 75-horsepower motor.

The cost for sorting and waste disposal per ton of material delivered to the waste dump for the past few years is as follows:

	1926	1927	1928	1/1929
Total tonnage	2,179,404	2,428,126	1,922,949	1,376,203
Coarse crushing	$0.0304	$0.0303	$0.0333	$0.0331
Sorting	.0420	.0388	.0394	.0334
Screening	.0042	.0039	.0015	.0007
Waste disposal	.0414	.0424	.0432	.0614
General operation and maintenance	.0113	.0134	.0182	.0190
Total	$ .1293	$ .1288	$ .1356	$ .1476

1/ First nine months.

The increased cost in 1929 is due to charging to operating expense the entire cost of erecting a new distributing station during the year.

FINE GRINDING

The material from the sorted-ore bin is drawn out through plain drop-lip chutes and fed into three types of crushers, that is, two 6-D Gates gyratories, one 6-inch Traylor Bulldog finishing gyratory, and two 4-foot Symons cone crushers. The 6-inch Bulldog finishing crusher has too small a feed opening for the size of feed handled and requires constant attention to avoid slabs bridging over. The Symons cone crushers have proven very satisfactory and are used exclusively, the others being used when repairs are being made to one of the cone crushers. These crushers handle about 125 tons per hour when set at 1 inch. A typical screen analysis of feed and product is:

Feed			Product		
Size	Per cent	Cumulative per cent	Size	Per cent	Cumulative per cent
Plus 5 inch	7.4	7.4	Plus 2 inch	13.8	13.8
4 "	19.1	26.5	1-1/2 "	23.7	37.5
3 "	30.8	57.3	1 "	28.6	66.1
2-1/2 "	17.1	74.4	1/2 "	14.5	80.6
2 "	10.9	85.3	3/8 "	3.9	84.5
1-1/2 "	7.1	92.4	1/4 "	3.4	87.9
1 "	7.6	100.	Minus 1/4"	12.1	100.0

The plus 1-inch material in the product is slabs that are 1 inch or less in thickness.

The 6-D gyratories and Bulldog finishing crusher are belt-driven by 50-hp. motors and the cone crushers by 100-hp. ball-bearing motors with Texrope drives.

The product of the cone crushers drops into a small receiving bin and is then conveyed without screening to the coarse-roll feed bin. The rolls are fed by small belt conveyors, one equipped with a Cutler-Hammer magnetic head pulley to protect the rolls from tramp steel. The product of the rolls is elevated by a series of inclined conveyors to the level of the ball-mill floor, where it is fed onto two double-deck Hummer screens. The upper screen has 7/8-inch square openings and the lower one is a Tyler No. 691 Ton-cap, which has rectangular openings 0.063 inch wide. The oversize of the 7/8 inch screen falls onto the same conveyor belt that carries the Symons cone-crusher product and returns with it to the feed bin. The product on the fine screen goes to the fine-roll feed bin, and the undersize is laundered to the concentrating floor. An average of 90 tons per hour is put through the 7/8 inch screen, and about 10 tons of this material passes through the fine screen. A circulating load of about 100 per cent is carried on this set of rolls.

The fine rolls are fed in the same manner as the coarse rolls, but there is no magnetic head pulley. The product of these rolls is carried on an incline conveyor to four 4 by 5 foot single-deck Hummer screens equipped with No. 691 Ton-cap screens. The undersize of these screens goes to the concentrating floor; and the oversize is returned by incline conveyors to the level of the roll feeder, where it can either be put in closed circuit through the rolls or all or any part put into a 14 by 7 inch bucket elevator and elevated to a conveyor over the tube-mill feed bins, where it is distributed by stationary trippers.

Two sets of rolls, Traylor AAA Ajo type, 24 by 60 inches, one for coarse crushing and one for fine, are in operation. Both sets of rolls are operated at 100 r.p.m., and each is driven by two 150-hp. motors with endless belts and idlers. Roll shells used are 8 inches thick; no grinding of shells is done, and only occasional burning off of the edges is required. The average life of a set of shells on the coarse rolls is 2,000 crushing hours, or a steel consumption of 0.102 pounds per ton, and on the fine rolls 2,800 crushing hours or 0.117 pounds per ton. A typical screen analysis of feed and finished product on the fine rolls is:

Feed			Finished Product		
Size	Per cent	Cumulative per cent	Size	Per cent	Cumulative per cent
Plus 1/2 inch	20.	20.	Plus 10 mesh	1.7	1.7
3/8 "	25.	45.	14	10.6	12.2
3 mesh	16.	61.	28	25.9	38.1
4	12.	73.	35	11.3	49.5
7	14.	87.	48	6.3	55.8
Minus 7	13.	13.	65	4.8	60.6
			100	6.5	67.1
			150	3.8	71.0
			Minus 150	28.9	100.0

An average of 40 tons per hour is finished on these rolls; the circulating load is about 200 per cent.

The mill proper is divided into 11 sections. Each is equipped with an 8 by 6 foot ball mill, operating at 22 r.p.m. and driven by a 225-hp. motor with endless belt and idler. The mills are lined throughout with cast manganese-steel liners and grates and carry a load of approximately 20,000 pounds of 6-inch forged chrome-steel balls. Each mill is served by two 30-inch Stephens-Adamson apron feeders equipped with revolution counters for calculating the tonnage handled. These feeders deliver the ore to the scoop box of the mill. A dilute pulp, about 60 per cent moisture, and 1 inch slot grates are used to get a quick discharge from the mill.

The discharge trunnion liner has a special designed flange to which is bolted a double trommel screen. The inner screen is made of 3/8 inch plate with 7/8 inch round holes and the outer one of 7-mesh screen with 0.089-inch square openings. The purpose of the inner screen is to carry the weight of the whole screen, keep the coarse rock off fine screen surface, and act as a feed distributor.

The undersize of the fine screen is run in a launder to the concentrating floor. The oversize of both screens is combined and returned to the scoop box of the mill by a 14 by 7 inch bucket elevator, or in a section having a tube mill this oversize may, if desired, be dropped by gravity into the feed bin of the tube mill. The average circulating load when grinding is done in closed circuit is 135 to 150 per cent. The average tonnage finished through the 7-mesh screen for each mill is 15 tons per hour of grinding time. A typical screen analysis of the feed to the ball mills, oversize, and finished product is:

Feed			Circulating Load			Finished Product		
Size	Per cent	Cumulative per cent	Size	Per cent	Cumulative per cent	Size	Per cent	Cumulative per cent
Plus 2-1/2 inch	1.8	1.8	Plus 3/4 inch	15.9	15.9	Plus 10 mesh	4.3	4.3
2 "	10.4	12.2	1/2 "	7.9	23.8	14	9.5	13.8
1-1/2 "	19.8	32.0	3/8 "	8.8	32.6	20	10.7	24.5
1 "	24.8	56.8	1/4 "	9.2	41.8	28	10.0	34.5
3/4 "	8.7	65.5	3/16 "	29.2	71.0	35	9.3	43.8
1/2 "	9.6	75.1	7 mesh	.7	71.7	48	7.0	50.8
3 mesh	10.2	85.3	14	18.1	89.8	65	5.1	55.9
7	9.1	94.4	Minus 14	10.2	10.2	100	7.0	62.9
Minus 7	5.6	5.6				150	5.6	68.5
						200	4.0	72.5
						Minus 200	27.5	100.0

The average life of a set of manganese-steel grates and feed end liners is 5,200 grinding hours and that of the breast liners 6,200 hours. The steel consumption, based on the tons screened through the 7-mesh screens, is

		1927	1928
Manganese-steel breast liners---pounds	per ton	0.186	0.159
feed end	do.	.062	.057
grates	do.	.035	.036
Total liner consumption	do.	.283	.252
6 inch chrome-steel balls	do.	.955	.927
Power consumption--Kw. h.	per ton	9.462	9.302

Originally there was one 6 by 12 foot tube mill in each section of the mill, but six of these have been removed, three to provide space for the two sets of rolls and three to make room for a boiler shop. The five mills in operation have been converted to operate as ball mills. These are operated at 25 r.p.m. and driven by a 150-hp. motor with endless belt and idler. Cast manganese-steel breast liners are used, but the grates and feed end liners are cast steel made locally in the company's own foundry. The usual ball load is 20,000 pounds. Balls used are the worn balls from the ball mills and some new 5-inch cast-steel balls. The general practice when a ball mill is stripped for relining is to send part of the old ball load to the tube-mill floor. Balls are fed to the mills daily in proportion to the running time of each mill. The material handled by these mills consists of fine-roll oversize, ball-mill oversize, or a combination of the two. It is fed into the scoop box of the mill by a gravity chute so arranged that the material feeds in as fast as it is taken out by the scoop. A dilute pulp, about 60 per cent moisture, carries the material rapidly through the mill. The grates have 1/2 inch slot openings. Trunnion trommel screens, similar in design to the ball mills, are used. The undersize of the fine screen goes to the concentrating floor, and the oversize of both screens is returned to the

scoop box by an 8-inch belt drag. A circulating load of 75 to 100 per cent is maintained. The average life of manganese-steel breast liners is 2,700 grinding hours, of cast-steel grates 1,700 hours, and of feed-end liners 1,800 hours. The consumption of steel per ton finished is:

Manganese-steel breast liners---pounds per ton		0.856
Local cast-steel feed end liners---pounds per ton		.060
Local cast-steel grates	do.	.091
Total liner consumption	do.	1.007
Ball consumption	do.	1.354
Power---Kw.h., per ton		7.640

An average of about 11 tons per hour is finished through the 7-mesh screens. A typical screen analysis of feed and product is:

Feed			Product		
Size	Per cent	Cumulative per cent	Size	Per cent	Cumulative per cent
Plus 3/4 inch	8.0	8.0	Plus 10 mesh	3.8	3.8
1/2 "	5.5	13.5	14	8.7	12.5
3/8	7.8	21.3	28	19.8	32.3
1/4	10.9	32.2	35	11.7	44.0
3/16	36.6	68.8	48	6.9	50.9
7 mesh	1.9	70.7	65	5.7	56.6
14	20.7	91.4	100	10.5	67.1
Minus 14	8.6	8.6	150	2.7	69.8
			Minus 150	30.2	100.0

CONCENTRATING

Each of the 11 sections in the concentrating department has the following equipment: One classifier, two V-tanks, two feed distributors, and eight Deister Simplex sand tables. The classifier is a 1- spigot Deister cone remodeled to suit local conditions. The feed distributors are of local design and consist essentially of two concentric cast-iron pots. The pulp drops through a pipe into the center of the inner and shallower pot, which overflows into the outer one, having a series of holes uniformly spaced. These feeders give fairly uniform distribution and require very little attention or upkeep. The Deister tables, as originally installed, had the old open type of head motion and "drum-stick" underconstruction. These were very unsatisfactory and have all been replaced with Deister Plat-O self-oiling head motion and slipper-bearing undercomstruction and have proved quite satisfactory. Covering for the deck under the riffles is 1/16-inch rubber sheeting with one ply of fabric in the center; linoleum is used for the apron. The rubber lasts several times as long as the linoleum, especially with the coarse feed. Spruce riffles 1 inch wide and 3/8 inch high are spaced 1 inch apart. The tables are belt-driven and operate at 267 r.p.m. with a stroke of 3/4 to 7/8 inch.

The undersize of the 7-mesh screens from all sources in each section passes through the classifier, which is placed in the launder leading to the V-tanks. This classifier makes a rough separation of the coarser material. The discharge of the classifier is laundered to a distributor, which feeds the first four Deister tables of the section. The overflow passes on to the V-tanks, where it is thickened. The overflow of the tanks goes to the tailings flume and the thickened product to the second distributor, which feeds the last four Deister tables of the section. Approximately 52 per cent of the material goes to the first four tables and 48 per cent to the last four. These tables all make three products, rough concentrates, dirty middlings, and tailings to waste. Their primary function is to produce clean tailings, without regard to the grade of the concentrates or middlings--the reverse of ordinary table work. The middlings are taken from between two definite points on the table, regardless of the character of material that may lie between these points; concentrates constitute all the remaining material produced, regardless of appearance or condition. Virtually the same amount of concentrates and middlings is cut, whether the tables are working at normal maximum or normal minimum loads. Each Deister table produces approximately 0.75 ton of concentrates and 6 tons of middlings per 24 hours. The concentrates consist of nearly all the freed gold and sulphides, plus coarse sands and true middlings. The middlings consist of a small amount of freed sulphides and a large amount of pulp containing some true middlings. This material is run in launders directly to the Wilfley floor and becomes the feed for two No. 6 Wilfley tables in each section. The functions of these tables are not only to recover the freed sulphides in this material but also to recover relatively high-grade coarse sands in large enough quantity to supplement the pulp in the re-treatment circuit up to the capacity limit of the fine-grinding equipment in that department and to produce tailings to waste. A minute cut of the freed sulphides is made from each table and run in launders to the reconcentrating tables on the re-treatment floor, and the middlings cut--about 1 ton from each table per 24 hours--goes to the sump.

RE-TREATMENT

The rough concentrates from all the Deister tables are run in launders to a pair of small trommel screens on the re-treatment floor which screen out any tramp oversize and send it to the sump. The undersize of the trommels is handled by a 2-inch Krogh centrifugal pump and is delivered to a dewatering and distributing tank, which feeds three Deister reconcentrating tables on the Wilfley floor. These tables produce a finished concentrate. Two concentrates are made, one a high-grade gold cut containing as much of the freed gold as possible and the other a shipping concentrate, which is run in launders to the tanks containing concentrates. A middlings cut, consisting mostly of freed sulphides, pyrrhotite, and pyrite with some galena and freed gold, is sent to two Deister reconcentrating tables on the re-treatment floor, while the reject from the tables is sent to the sump. The purpose of retabling the middlings cut a second time before regrinding is to remove as much freed gold and galena as possible from this material before it is reground. These tables make high-grade gold concentrates and shipping concentrates, and the reject is sent to the sump.

The material from the sump is the feed for the re-treatment department proper. It is elevated by 14 by 7 inch bucket elevators to a dewatering box, which feeds the partly thickened material to one of two 5 by 5 foot Allis-Chalmers ball mills equipped with shovel-wheel feeders. The overflow of the

dewatering box and shovel wheel, together with the product from the mill, is pumped by a 4-inch Krogh centrifugal sand pump to a V-tank on the concentrating floor. The partly thickened product from this tank is run in launders to two 4-spigot Deister cone classifiers. The product from each spigot is tabled on one of eight Deister re-treatment tables. The functions of these tables are to recover as much freed gold and galena as possible from this material in a low-grade product and to produce a low-grade reject. Three products are made on these tables, low-grade concentrates, middlings which are returned to the sump, and a reject which formerly went to waste but is now sent to the flotation circuit. The low-grade concentrates from the re-treatment tables are laundered to a Deister reconcentrating table on the re-treatment floor. This table makes high-grade gold concentrates, shipping concentrates, and a reject, which is sent to the sump.

The overflow from the V-tank and classifiers is pumped by a 3-inch Byron-Jackson slime pump to a second V-tank. The thickened product of this tank goes to two Deister slime tables and the overflow to waste. The slime tables produce shipping concentrates, middlings to the sump, and a reject which formerly went to waste but now goes to the flotation circuit.

The feed for the flotation circuit is pumped by a 3-inch Byron-Jackson slime pump to a double V-tank, the overflow of which goes to waste and the product to a 2- cell 18-inch Minerals Separation Sub-A flotation machine. This machine produces a shipping concentrate from the first cell and tailings to waste. The flotation plant is still in the experimental stage, and more cells may be added. At present about 1 ton of concentrates, assaying $75.00 in gold, 10 to 14 per cent lead, 4 per cent zinc, and 40 to 45 per cent iron, is being produced per 24 hours from 80 to 100 tons of feed assaying $1.50 to $2.00 in gold, 0.1 to 0.2 per cent lead, 0.5 per cent zinc, and 20 per cent iron. Potassium amyl xanthate and cresylic acid are the flotation reagents used.

The above description has pointed out that the table work in the Alaska Juneau mill is quite distinct from that in the ordinary concentration mill. The concentrates and middlings cuts are both enormously large in proportion to the mineral content in the feed, which is extremely small. An average of 1.15 pounds of lead is recovered per ton of ore fine ground. It will be noted that all these tables making clean tailings do not make a finished product, and those tables making a finished product do not make tailings.

Numerous experiments have been made with finer grinding and better classification of table feed, but the additional recovery has not been found to be great enough to cover the increased cost of operation. The typical distribution of values in the Deister and Wilfley table tailings is given in the following table:

Size	Deister	Wilfley
Plus 28 mesh	$0.2766	$0.3063
35	.2747	.3370
48	.2734	.3504
65	.2575	.3335
100	.2351	.2928
150	.2170	.2861
200	.2265	.2714
Minus 200	.2493	.3161

All fine tailings to waste are discharged into one of two 42 inch tailings flumes and carried to the beach, a distance of about 3,000 feet. The grade of the old flume is 4 per cent and the new one 2-1/2 per cent. A dilute pulp, 10 per cent solids, is required on account of the high percentage of coarse sand in the tailings. A typical screen analysis of tailings is

Size	Per cent	Cumulative per cent
Plus 10 mesh	4.3	4.3
14	9.9	14.2
20	16.8	31.0
28	13.6	44.6
35	13.8	58.4
48	9.5	67.9
65	5.7	73.6
100	6.0	79.6
150	3.4	83.0
200	6.3	89.3
Minus 200	10.7	100.0

The old flume is lined with concrete. The new flume has 2-inch cast-iron liners cemented in place and a few hundred feet of rubber lining. These have not been in use long enough to allow comparisons of their relative wearing value.

Shipping concentrates produced on the re-treatment floor are caught in boxes at the end of the tables and packed in double burlap sacks. Concentrates produced on the Wilfley floor are run in launders to the concentrates tanks with filter bottoms. The concentrates, after draining in the tanks, are shoveled onto the floor, allowed to dry for 2 or 3 days, and then packed in paper-lined burlap sacks. The average moisture content is about 5 per cent. These concentrates average about $350.00 per ton in gold, 35 ounces silver, 60 per cent lead, and 20 per cent iron. The high gold content of these concentrates is due to very thin flat particles of gold, which tend to ride on the galena. The ratio of concentration after fine grinding has been about 1000 to 1, but since flotation concentrates are being made this has dropped to 750 to 800 into 1.

The high-grade gold concentrates, assaying $20,000 to $30,000 a ton in gold, are collected in an amalgam barrel for treatment. Six hundred ounces of quicksilver, which has been cleaned with a 10 per cent nitric acid solution and about 3 pounds of freshly slaked lime, is added to the charge and all ground for 5 or 6 hours. The amalgam barrel is of cast iron, is 2 by 4 feet in size, revolves at 30 r.p.m., and has one piece of 6-inch shafting as a grinding medium; the shafting keeps the inside of the barrel smooth and makes it easier to clean out all the amalgam after each run. The barrel is discharged while running through a 1/2-inch hole into a 4-foot diameter mechanical batea which removes most of the galena. The residue in the batea is carefully cleaned by hand, and the cleaned amalgam is squeezed in an air-operated press. The value of the concentrates discharged from the batea averages about $75.00 in gold.

The amalgam is retorted at regular intervals in an oil-fired cast-iron horizontal retort. The sponge-gold recovery is 60 to 65 per cent of the weight of the amalgam and the mercury loss about 1 per cent. The sponge gold is melted in an oil-fired tilting furnace with a No. 10 graphite retort for a crucible. Soda and borax glass are used for flux, and the molds are heated and oiled with boiled linseed oil. The bullion is sampled while molten and before pouring by a graphite mold made from an old retort. One bar, 1200 to 1600 ounces in weight, is melted at a time. The average grade of the bullion is gold, 850 fine; silver, 130 to 135 fine. Approximately 80 per cent of the total gold recovery is in the form of bullion. One man takes care of amalgamating, retorting, and melting.

The operating crew for the mill, outside of the coarse crushing and sorting plant, is 1 shift boss and 11 men for each shift. A clean-up and repair crew takes care of general cleaning up, handling of concentrates, maintenance and repairs, etc.

The company has a large dock and warehouses where ocean-going steamers land all supplies and load concentrates for shipment to the smelter. Supplies are delivered to the mill by an inclined tram. The ball and tube mill floors are each equipped with a 50-ton Shaw electric traveling crane, and the crushing plant has a 20-ton crane of the same make. Supplies on other floors are handled on monorail tracks.

Table of Assays and Recoveries

Year 1927	Weight		Gold	
	Tons	Per cent	Assay, dollars per ton	Per cent of total
Rock trammed	4,267,810	100.00	$0.7743	100.00
Coarse table tailings	2,428,115	56.89	.1900	13.96
Coarse table middlings reground	1,839,695	43.11	1.5455	86.04
Bullion	- - -	- - -	.9659	53.77
Lead concentrates	1,420	.03	.2937	16.69
Fine tailings	1,838,275	43.08	.2801	15.58
Total tailings	4,266,390	99.97	.2288	29.54
Total recovery	- - -	- - -	1.2656	70.46
Year 1928				
Rock trammed	3,718,140	100.00	1.1062	100.00
Coarse table tailings	1,922,949	51.72	.2097	9.8
Coarse table middlings .... reground	1,795,191	48.28	2.0666	90.20
Bullion	- - -	- - -	1.3424	58.59
Lead concentrates	1,947	.05	.4071	17.17
Fine tailings	1,793,244	48.23	.3174	13.84
Total tailings	3,716,193	99.95	.2617	23.64
Total recovery	- - -	- - -	1.7496	76.36
Year 1929[1]				
Rock trammed	2,904,560	100.00	1.1007	100.00
Coarse table tailings	1,376,203	47.38	.1919	8.26
Coarse table middlings reground	1,528,357	52.62	1.9190	91.74
Bullion	- - -	- - -	1.2724	60.82
Lead concentrates	1,627	.06	.3699	17.69
Fine tailings	1,526,730	52.56	.2770	13.23
Total tailings	2,902,933	99.94	.2367	21.49
Total recovery	- - -	- - -	1.6423	78.51

1 First 9 months.

Summary of All Production per Ton

	Milled	Mined	Milled	Mined	Milled	Mined
Gold	$1.2656	$0.5456	$1.7496	$0.8447	$1.6423	$0.8642
Silver	.0187	.0081	.0252	.0122	.0233	.0123
Lead	.0542	.0233	.0709	.0342	.0841	.0442
	1.3385	.5770	1.8457	.8911	1.7497	.9207

The company maintains a foundry at Treadwell, the site of the famous Treadwell mine, just across the channel on Douglas Island. The foundry is equipped with a Heroult 2-ton electric furnace and makes most of the necessary castings, pulleys, gears, liners, etc.; an annealing furnace has been put into operation, and alloy-steel liners will be made. Scrap from the mill and from the old Treadwell plants is utilized by the foundry. A machine shop and a boiler shop are located in the mill building and are equipped to make any necessary repairs.

MILLING COSTS

The costs for the various stages of fine milling for the last few years, based on the total tonnage of fine milled, are:

	1926	1927	1928	1/ 1929
Tons milled	1,659,791	1,839,686	1,790,491	1,528,357
Coarse crushing	$0.0304	$0.0303	$0.0333	$0.0331
Sorting	.0094	.0087	.0122	.0122
Screening	.0063	.0077	.0025	.0025
Intermediate crushing	.0206	.0143	.0190	.0194
Fine grinding	.1830	.1976	.1606	.1604
Concentrating	.0165	.0226	.0221	.0225
Re-treatment	.0178	.0175	.0179	.0147
General operation and maintenance	.0449	.0530	.0588	.0514
Total	.3289	.3517	.3264	.3162

1/ First 9 months.

The cost per ton handled in each operation, based on the tonnage actually handled in that operation, is:

	1927	1928	1/ 1929
Coarse crushing	$0.0303	$0.0333	$0.0331
Sorting	.0388	.0394	.0534
Coarse waste disposal	.0424	.0432	.0614
Screening, coarse	.0080	.0030	.0024
Intermediate crushing	.0551	.0739	.0686
Screening, hummer	.0063	.0089	.0130
Ball mills	.1988	.1613	.1609
Rolls	.1100	.0738	.0806
Tube mills	.2017	.1773	.1593
Concentrating	.0226	.0221	.0225
Re-treatment	.1972	.1974	.1732
General mill, operating maintenance, etc.	.0305	.0378	.0361
Total	.2257	.2285	.2363

1/ First 9 months.

Milling costs for power, labor, and supplies based on tonnage trammed are distributed as follows:

	1927	1928	1929
Power	$0.0321	$0.0272	$0.0288
Labor	.0984	.1085	.1081
Supplies	.0952	.0928	.0994
Total	$ .2257	$ .2285	$ .2363

The cost and tonnage handled per man per day for sorting and milling are:

	1927	1928	1929
Sorting:			
Crew (direct)	79.47	71.61	72.35
Cost per man per day	$ 8.53	$ 8.26	$ 8.91
Tons rejected per man per day	83.71	73.37	69.67
Milling:			
Crew (direct)	110.46	106.53	110.34
Cost per man per day	$17.75	$16.24	$16.94
Tons milled per man per day	46.04	46.04	50.74

The outstanding features of the operations of the Alaska Juneau are: An ore body of such size and character as to permit a caving system of mining whereby ore is mined and delivered to the mill at a cost of $0.26 to $0.29 per ton; rejection by screening and hand-sorting of approximately 50 per cent of the material mined at a cost of $0.13 to $0.15 per ton rejected; and milling of the sorted ore at a cost of $0.31 to $0.33 per ton milled, giving a total operating cost of $0.51 to $0.57 per ton trammed from the mine.

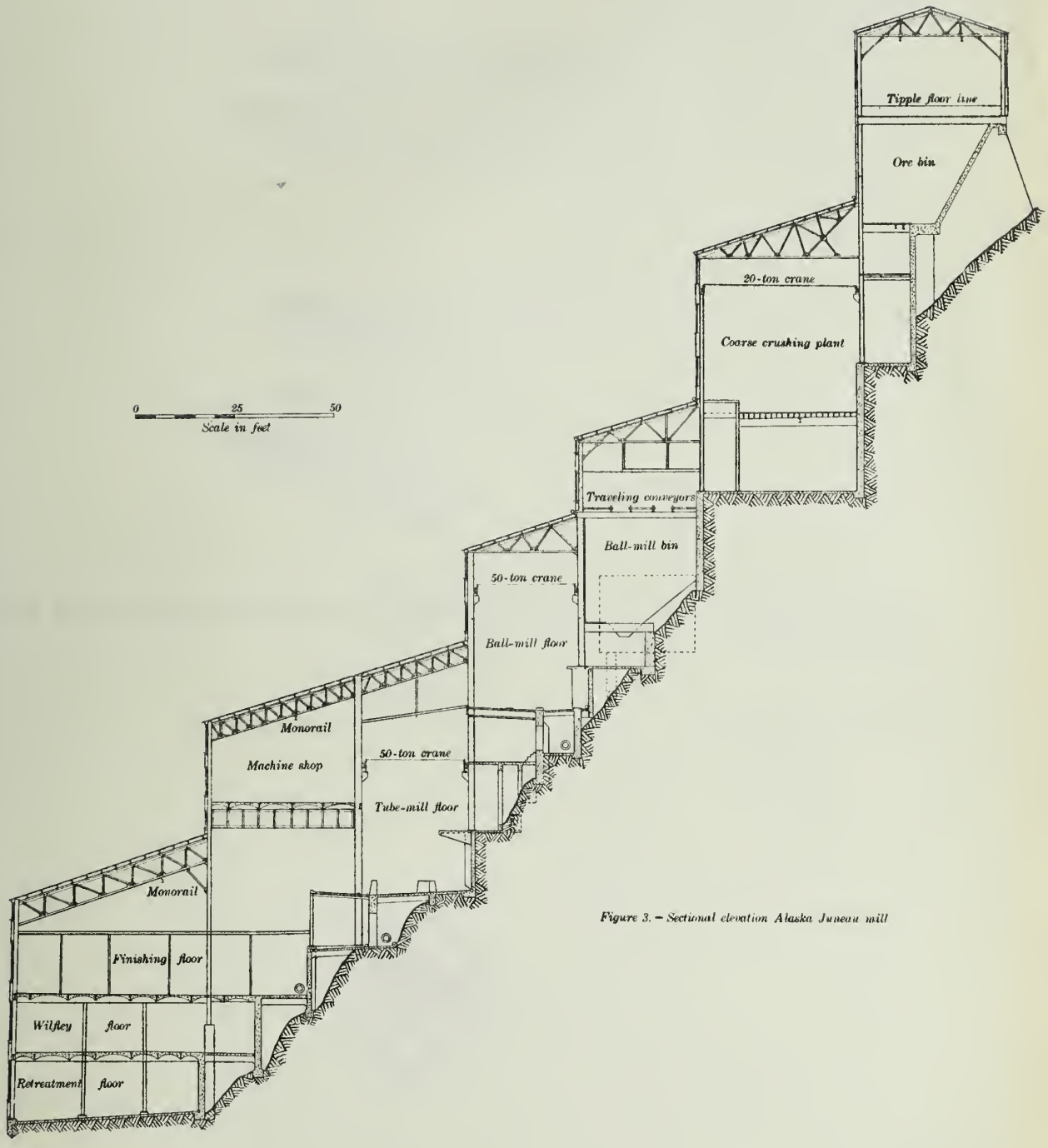
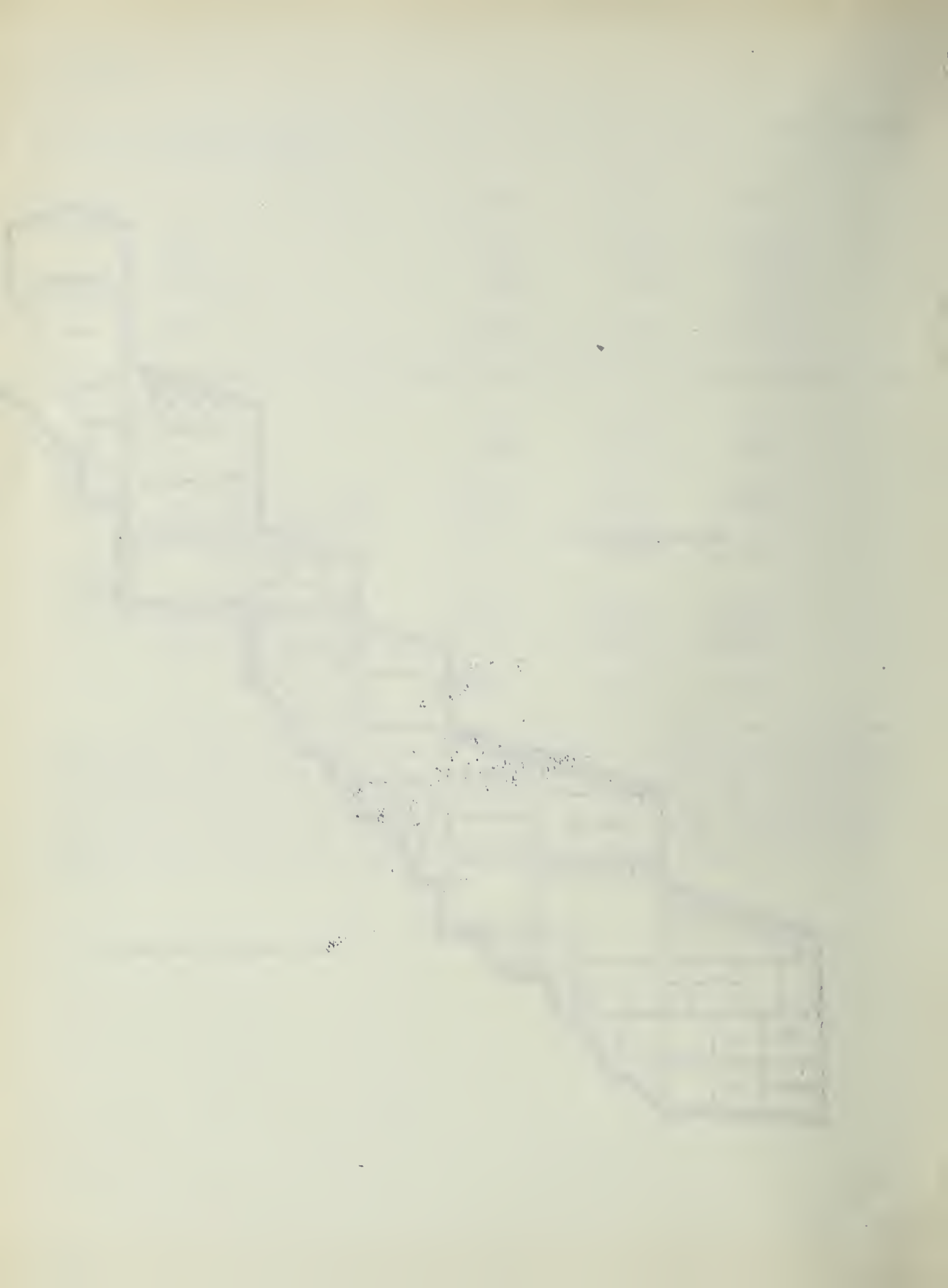


Figure 3. – Sectional elevation Alaska Juneau mill



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MINING METHODS AT THE OLD DOMINION MINE, GLOBE, ARIZ.

BY

A. H. SHOEMAKER

I. C. 6237
February, 1930

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AT THE OLD DOMINION MINE, GLOBE, ARIZ.[1]

By A. H. Shoemaker[2]

INTRODUCTION

This paper describing the mining practices at the Old Dominion mine, Globe, Ariz., is one of a series being prepared by the United States Bureau of Mines on mining practices, methods, and costs in the various mining districts in the United States.

The Old Dominion mine is situated in the eastern part of the Globe-Miami district, about 1-1/2 miles north from the center of the town of Globe. An average of nearly 1,200 tons per day was mined during 1928. The present production is about 1,500 tons per day; of this amount, approximately 100 tons is direct-smelting ore that is shipped to the smelter at Miami and 1,400 tons milling ore which is treated in the company's concentrator at Globe. About 600 men are employed at the mine.

ACKNOWLEDGMENTS

The author acknowledges the assistance of H. A. Leidich, mine superintendent, and E. A. Wilske, assistant chief engineer of the Old Dominion Co., and of E. D. Gardner, supervising engineer, C. H. Johnson, assistant mining engineer, and G. E. McElroy, mining engineer, of the United States Bureau of Mines Southwest Experiment Station at Tucson, Ariz.

HISTORY

The first location, on what now comprises Old Dominion ground, was made in 1875. Due to the lack of transportation, interest in mining was at first confined to the rich silver ores in the district, and it was not until 1881 that the first copper was mined. The Old Dominion mine has been operated since that time with very few interruptions.

The present company, which is a consolidation of the Old Dominion Copper Mining and Smelting Co. and the United Globe Mines, was formed in 1917.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6237."

2 One of the consulting engineers, U. S. Bureau of Mines.

GEOLOGY

The geology of the area about Globe has been described in detail by the U. S. Geological Survey.[3] This section consists of about 2,300 feet of limestones, quartzites, and shales intruded by thick sills of diabase and overlaid in places by a thick dacite flow and by widely varying thicknesses of the Gila conglomerate.

The Old Dominion vein, in which the principal ore bodies of the mine have been found, is a true fissure vein from 2 to 60 feet wide. The average dip is about 60°, but varies from 45° to vertical. The ore bodies in the vein occur as localized shoots. Generally they have a flat rake to the west.

In the oxidized zone there are many comparatively small flat-lying ore bodies closely associated with the Old Dominion vein and occurring as replacements of particular highly-altered beds in limestone.

These ore bodies are generally capped by a heavy gossan. The commonest form of alteration of the limestones in which these ore bodies occur is to a kaolinite mass that quickly becomes heavy upon the beginning of mining.

Several small outside veins are also mined. The dips of these veins range from 35° to vertical. The walls are quite good, so that open-stope mining methods are applicable.

On the lower levels an ore body of considerable importance occurs in a diabase sill as a wide mineralized zone without definite structural boundaries.

There is an extremely wide variation in the size and dip of the ore bodies, in the character of the ore, and in the nature of the wall rock. In the mine in the Old Dominion vein the footwall of the largest ore shoot is quartzite and limestone and the hanging wall is a very soft highly-altered diabase. The vein material on the footwall side is limestone or quartzite with added silica and sulphides; on the hanging-wall side it is leached altered diabase. The hanging-wall side of the vein has a higher pyrite but lower copper content than the footwall side. Considerable granular pyrite occurs associated with the copper sulphide minerals in many places in the main vein. The ore that contains a large proportion of pyrite is difficult to hold in place after stoping begins.

In the extreme east end of the mine the vein is nearly vertical with good quartzite walls; these conditions are ideal for cut-and-fill and shrinkage stoping.

The normal sequence of oxidized, enriched, and primary zones is found in the Old Dominion vein. In the oxidized zone chrysocolla is the most abundant mineral. There are minor amounts of malachite, native copper, and cuprite. Chalcocite occurs as a thin coating on pyrite in the enriched zone. Chalcopyrite is by far the most important primary ore mineral. Bornite in places is abundant, particularly at the horizon of the lowest limestone.

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3 Ransome, F. L., The Copper Deposits of Ray and Miami, Ariz. U. S. Geol. Survey, Prof. Paper 115, 1919, 192 pp.

Every section of the mine presents a different problem, requiring a special study; hence, no general picture can be given. The hardness of the ore and the physical condition of the wall rocks varies widely. In places the ground will stand without support and in others it is excessively heavy.

The grade of the ore varies from 2.0 to 12.0 per cent copper, the oxidized ores being of the highest grade.

METHODS OF PROSPECTING AND EXPLORATION

When a new level is opened, a complete plan of development is made by the geological department. The first operation in accordance with this plan is to run a drift in the footwall which is intended for subsequent use as an extraction drift. The aim is to place it about 40 feet from the footwall, unless excessively heavy ground is expected, in which case the distance may be increased up to 70 feet. It does not pay, however, to place an extraction drift at too great a distance from the vein, for the added footage in crosscutting and the consequent inconvenience during stoping operations may more than balance the repair bill due to heavy ground; also, the heavy ground may extend a considerable distance from the vein, in which case there would be no advantage in moving out the footwall drift.

Crosscuts are run through the vein from the footwall drift at 240-foot intervals. This is sufficiently close in most cases to prove or disprove the occurrence of commercial ore. Closer development for mining purposes is done later. After the ore bodies have been proved, a drift is generally run on the footwall of the vein. Prospect raises are then driven on the footwall, spaced at such intervals as will best suit the mining method.

In searching for bedded ore on the west side of the mine, a different plan is followed. Crosscuts and raises in possible ore-bearing areas are run close enough together to make sure that no important ore bodies are missed. The interval necessary to insure this result is much smaller than the regular crosscut or raise interval. No definite distance is used, as each case requires a different treatment.

The aim is to make the preliminary development serve in stoping. In the largest number of places this is possible. However, when developing an entirely new area, some preliminary work must be done before a probable mining method can be outlined.

METHODS OF SAMPLING AND ESTIMATION

Sampling is done under the direction of the geological department. All development faces are sampled at 5-foot intervals with a prospector's pick. Where the ore is sufficiently high grade to leave no doubt that it can be profitably extracted, this pick sampling is taken as final. If the material is marginal, the original sampling is checked either by channel or drill samples.

Careful pick sampling quickly and cheaply outlines those areas in which the grade of ore must be more carefully determined. A considerable amount of expensive channeling can be avoided in this way.

A carefully worked out set of rules is used for estimating ore. The common end-area formula is used for the calculation of tonnages.

The formula is as follows:

$$V = \frac{L}{2} \quad (A_1 + 2A_2 + 2A_3 + \_\_\_\_\_ A_n)$$

Where V is volume, L the distance between sections, A_1, the area of first section, A_2, the area of second section, A_3, the area of third section, and A_n, the area of Nth section.

An average factor of 12-1/2 cubic feet of ore in place is taken as 1 ton of ore.

Ore is divided into two classes - developed and prospective. Developed ore consists of ore that has been cut on two different levels and has been proved to be continuous between by one or more raises. If ore has been cut on a level only by a crosscut, provided there is proof of continuity to another level, the ore for a distance of 60 feet on either side of the cross-cut is considered as developed ore. This is an arbitrary figure, based upon past experience which has shown that 60 feet is the probable minimum lateral extent of the ore. Prospective ore is that which has been cut only on one level and must be given an assigned altitude. The altitude given varies with the section of the mine, but it is never over 100 feet.

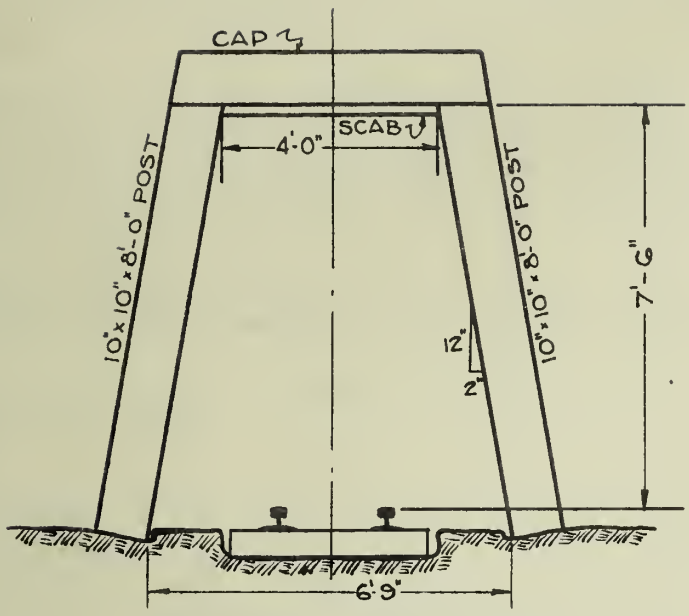
The mine is divided into 100-foot blocks. For each block on every level, estimate cards are made out indicating the class, grade, and tonnage of ore. This is a great convenience, making it possible to obtain estimate figures for particular areas by simply consulting the files.

In addition to these estimates, one is made of the total tonnage available in each stope, together with the estimated rate of mining. This gives the operating department data on when new sections must be ready for mining to insure a uniform production.

METHODS OF DEVELOPMENT AND MINING

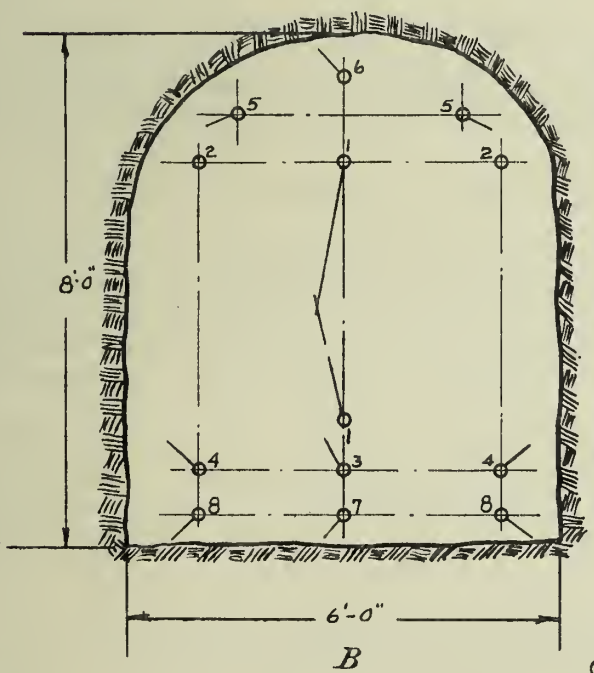
The mine is operated through one vertical shaft (A), consisting of one pump and two skip compartments and two compartments for handling men and supplies. Three other shafts (C, Kingdon, and Grey) connected to the lower levels are maintained for ventilation and emergency exits. A one-compartment shaft (Pump) is used solely for handling water.

Haulage levels are run at 200-foot vertical intervals. Intermediate levels for mining are nearly always run midway of the haulage levels.

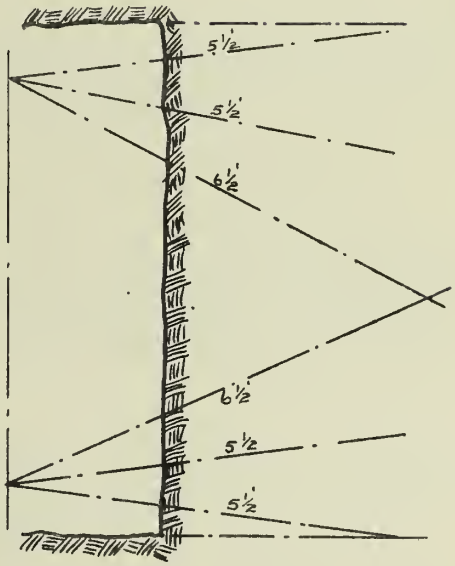


STANDARD DRIFT SET

A



B



STANDARD ROUND FOR
6"x8" DRIFT- 13 HOLES
ORDER OF FIRING AS SHOWN

FIGURE 1



There is no uniformity in the spacing of raises, as the requirements for different mining systems vary widely.

Drifts

Two sizes of drifts are run: one for motor haulage and the other for hand-tramming.

Untimbered motor-haulage drifts are 6 by 8 feet in the clear. In timbered haulage drifts 8-foot posts with a 4-foot spreader are used. Where a ditch is needed, one 9-foot post is used. Figure 1, A, shows the standard timbered motor drift. Table 1[4] gives the number of sets and data for placing the timber for different curvatures of track.

Untimbered hand-tramming drifts are 5 by 7 feet. Posts 7 feet 3 inches long with 3-foot 6-inch spreaders are used when these drifts are timbered.

A standard round is used for all development work. Figure 1, B, shows the standard drift round.

Both mounted and unmounted drills are used in drifting. Most of the quartzite and some of the diabase requires the use of a mounted drill. In drifting, the unmounted drill is supported by a hook on a piece of steel. Jackhammers are used in sinking; hand-rotated stopers in raising and stoping. One and one-quarter inch hollow-round steel is employed with the mounted drifters and 1-inch hollow-round steel with the unmounted machines. The smaller steel with same-sized bits is also used with pluggers and stopers.

The steel changes differ by a length of 1 foot. The gauge of starters made of the larger steel is 2 inches and of the smaller steel 1-3/4 inches. A difference of one-sixteenth inch exists between changes for both sizes of steel. The standard double taper of 5 and 14° is used for all bits.

Air pressure at the working faces is 80 pounds per square inch.

Forty per cent strength gelatin dynamite is used in practically all the work in the mine. Occasionally 60 or 80 per cent is used in the cut holes of shaft rounds. All primers are made by inserting the detonator in the side of the cartridge and tying it securely with a string. One stick of powder is placed in the hole before the insertion of the primer.

Electric blasting in all development work is compulsory. In some instances electric blasting is required in stoping, although fuse and blasting caps are generally used. Rounds are blasted from a 250-volt mine circuit. Stemming in cartridge form is used in all rounds.

The depth of rounds broken in drifts averages about 4 feet. This varies, however, with the type of ground and the number of shifts per day. Where speed is required and three shifts are being worked in the same heading, the depth of a round depends upon the amount of broken material that can be handled by one shift in order to have the cycle of operations coincide with the shifts.

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4 Curve Data Table for Standard Timbered Motor Drifts. Eng. and Min. Jour., vol. 127, No. 26, June 29, 1929, p. 1043.

Where possible, each crew is given two headings, and drilling and shoveling are done on alternate shifts. In this case the rounds are drilled from vertical columns. In the primary development on the lowest level this is not practicable, so drilling and mucking are done simultaneously; the back breast holes and top cut holes are drilled from a crossbar. If the ground is not too hard the lifters are drilled with a plugger.

All mucking in development faces is done by hand. A few drifts on mining levels have been mucked with scrapers, but their use has been discontinued because the efficiency of this operation proved to be no greater than that of shoveling by hand. Experiments have been made with mechanical shovels in drifting, but the largest drift run is too small for the effective use of these machines. Again, the working conditions in primary development headings on the lowest levels, where speed is necessary, are so bad that it is almost impossible to keep a trained crew of operators.

In excessively heavy ground, booms are used to protect the shovelers. Head protection is always given in ground that requires timbering. If booms are unnecessary, lagging is pushed out over the cap of the last set and blocked tightly against the back.

Guniting has been used to excellent advantage where the ground is not heavy, but slacks on exposure to the air. It is particularly effective in badly brecciated quartzite. It is not effective where there is acid water. Hundreds of feet of timbering have been avoided by guniting.

Raises

A large percentage of the raising done is in the vein. As a consequence, a great deal of bad ground is encountered, so that miners of experience and exceptional skill are required.

With few exceptions, raises have two cribbed compartments. In the usual development raise each compartment is 3 feet 4 inches square in the clear and is an independent unit (fig. 2). This arrangement has a decided advantage in bad ground, for the manway compartment can be cribbed rapidly, the back secured, and then the chute compartment timbered in safety. Also, as the cribbings usually lap between the two compartments, the lining of the chute is not necessary while the raise is being driven. The ore chutes are lined after the raises are run if enough tonnage is to be passed through them to cause the cribbing to wear seriously.

Transfer raises are usually run with two cribbed compartments 4 by 4 feet in the clear. If the ground is good, as it usually is, the cribbing is taken out and reused; otherwise, the cribbing is left in and the raises lined. Five by eight-inch cribbing placed on the edge is used in all raises.

Air and water pipes are placed in all raises while they are being driven and are kept close enough to the back to require only one length of hose.

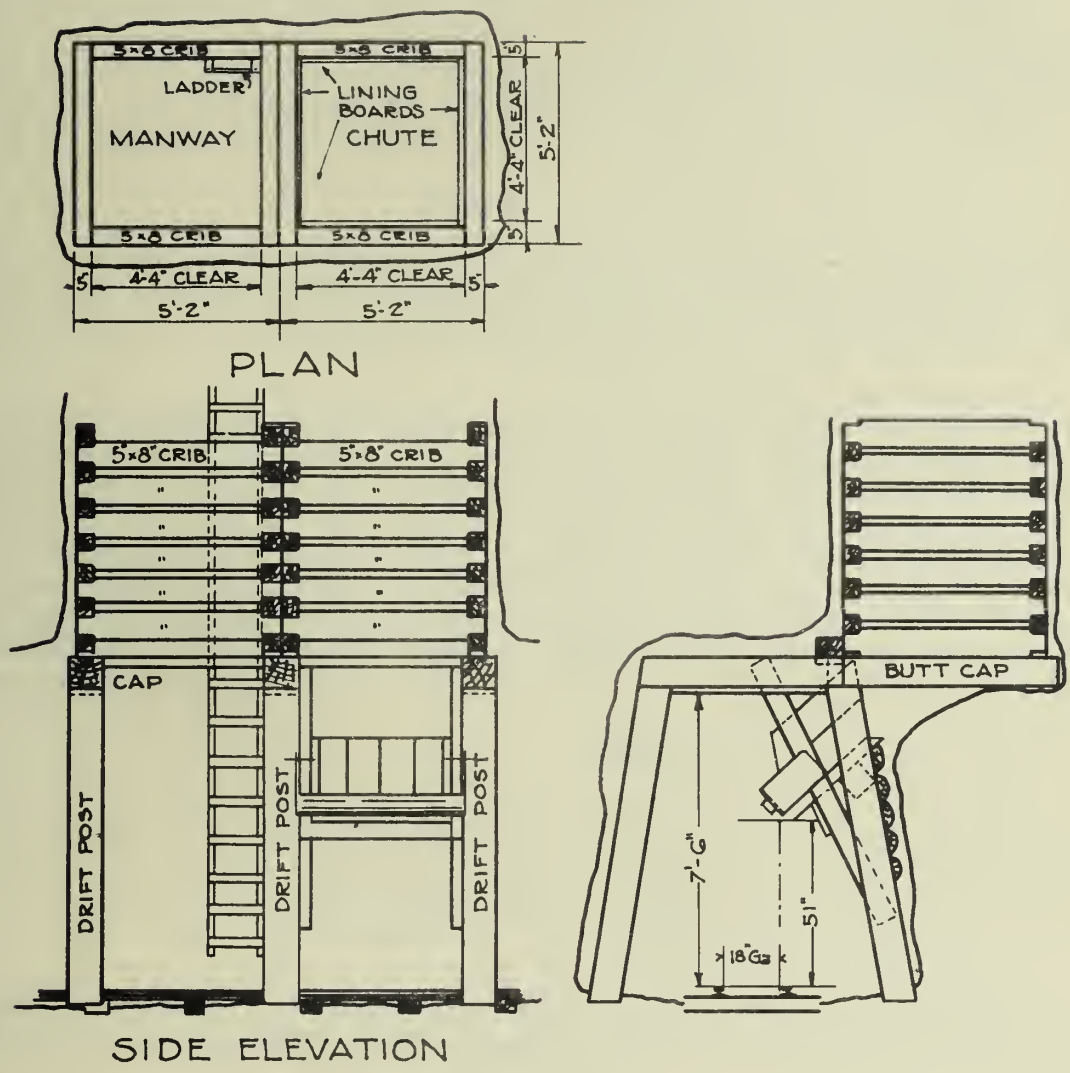


FIGURE 2.
TYPICAL CRIBBED RAISE



Before blasting in a raise, a bulkhead of 5 by 8 inch cribbing on end must be placed over the manway. Chutes are kept full, and only enough broken rock is drawn to take care of the material blasted in each round while the raise is in progress.

Shafts

The cage compartments in the operating shaft are 4 by 6 feet in the clear. The man cages have three decks with a capacity of nine men per deck.

The operative shaft is deepened while being used. An auxiliary hoist is placed in the pump compartment on the lowest level. Three compartments are sunk; the other two are raised through later.

Stoping

The wide variation in the size, dip, and grade of the ore bodies at the Old Dominion make it necessary to employ many mining systems. The factors influencing the choice of the mining method are size, dip, condition of wall rock, and grade of ore.

Whenever the walls and vein material stand well and the dip is 75° or more, a shrinkage method is used. Where the ore is high grade and clean mining is a necessity but the back requires support, top slicing is the method chosen. If the dip is 60° or more and the ore is so low grade that it can not be profitably extracted by selective mining, caving is used. Square-setting is used only where ore is high grade and the over-lying country must be supported. Open stopes are run on small veins where the walls are good and the dip is too flat for shrinking. At present (December, 1929) the percentage of ore from the various methods is as follows:

	Per cent
Caving	57.0
Square-setting ..	21.3
Shrinkage	14.1
Top slicing.....	3.8
Open stopes	3.8

These percentages, of necessity, vary; however, the list shows the complexity of the stoping problem of the mine.

No sorting of sulphide ore is done. Occasionally high-grade oxidized ore is hand-sorted in the stope.

Caving.--A caving method of mining, using an inclined slide, has been adapted for extracting comparatively small ore bodies in the Old Dominion vein. This system is a modification of the Morenci inclined slide, and credit for the original development and the first successful operation of the system belongs

to the Morenci branch of the Phelps Dodge Corporation.[5]

Considerable experimenting has been done at the Old Dominion, and several important changes have been made since the first trial stope. The general method is to divide the 100-foot interval between two levels into two lifts. If either of these lifts is above a level with no motor haulage, the ore is hand-trammed into transfer chutes run from the haulage level. Timbered pony drifts are run along the footwall of the vein. On the level, the pony drift is immediately over the footwall development drift. The pony drift for the top lift is a small intermediate drift. Transfer raises on 40-foot centers are run from the bottom pony drift to the upper one for removing the ore from the upper lift, which of course is mined first. From the pony drifts, timbered slides on 20-foot centers are run at an inclination of 36° from the horizontal to the hanging wall. A shrinkage stope is then run on the footwall to the top of the lift. After this is completed the remainder of the block is undercut and the ore drawn.

In the first stopes worked by this method inclined slides were run on the footwall each way from a central raise. From these slides crosscut slides were run to the hanging wall. This method placed the crosscut slides at different elevations, so that the drawing of the ore from the lower slide might collapse the upper one (fig. 3). The timbering of the cross slide was complicated and would not stand as much weight as when started from a flat pony set. Supplies had to be handled always on the incline.

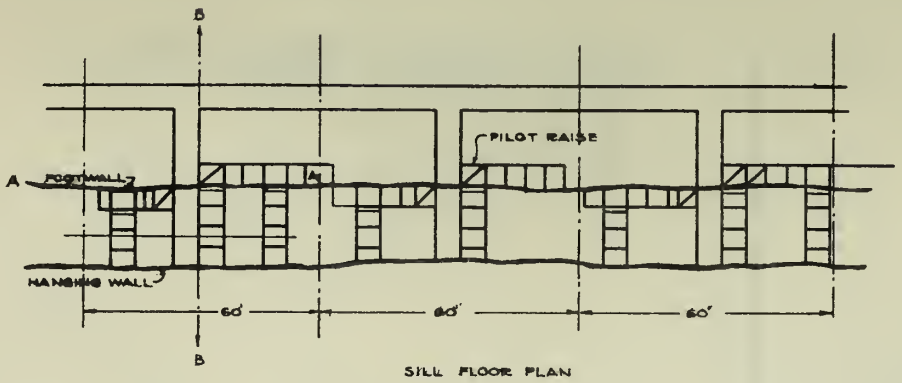
The driving of a flat pony drift on the footwall is now standard practice (fig. 4), as it does away with the objectional feature of the drawing from the lower slide collapsing the upper. It permits simple timbering to be used for the brow set of the cross slide and allows supplies to be handled with comparative ease.

Two lifts are taken in veins that are flatter than 75°. Experience has shown that the ore will not follow down the footwall when an attempt is made to draw a stope from level to level. The distance that the ore could be drawn in hard ground could safely be extended more than 50 feet, but as two lifts are necessary, both are made the same height.

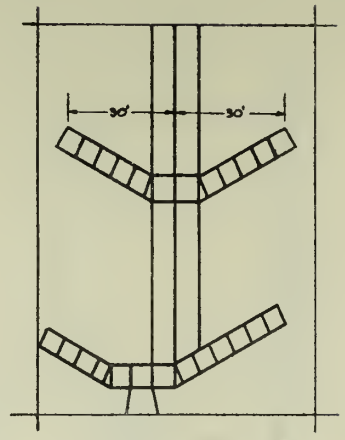
Many of the sections caved in this manner are small pillars left between old stopes. There is no uniform size for a caving stope. In one case, however, the operation has been made continuous over a 400-foot length of vein by keeping the advance development, shrinking, and undercutting in the proper sequence. The development of a lower lift can start just as soon as drawing from the upper slides is completed; drawing can start from the lower installation when the drawing from the upper installation is completed far enough beyond the lower so that drawing will not collapse a needed section of the upper pony drift.

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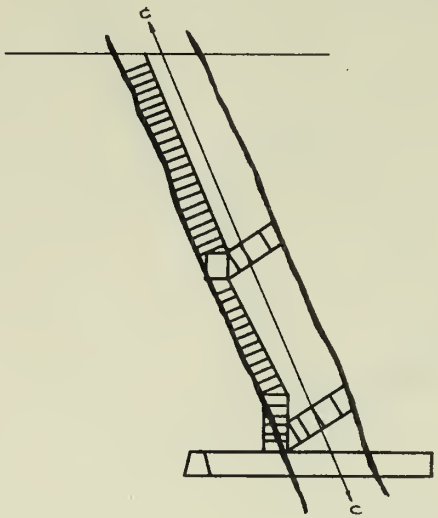
5 Mosier, McHenry, and Sherman, Gerald, Mining Practice at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz. I. C. 6107, U. S. Bureau of Mines, March, 1929, 33 pp.



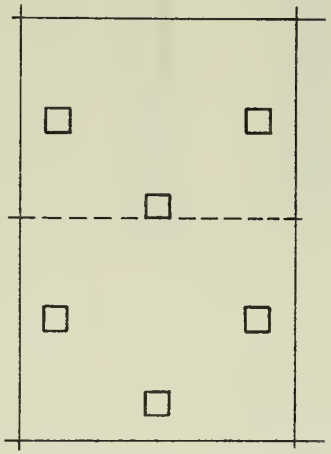
SILL FLOOR PLAN



SECTION THROUGH A-A



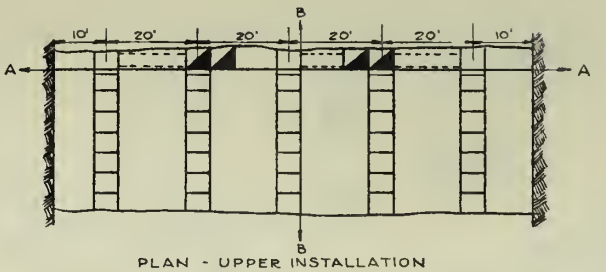
SECTION THROUGH B-B



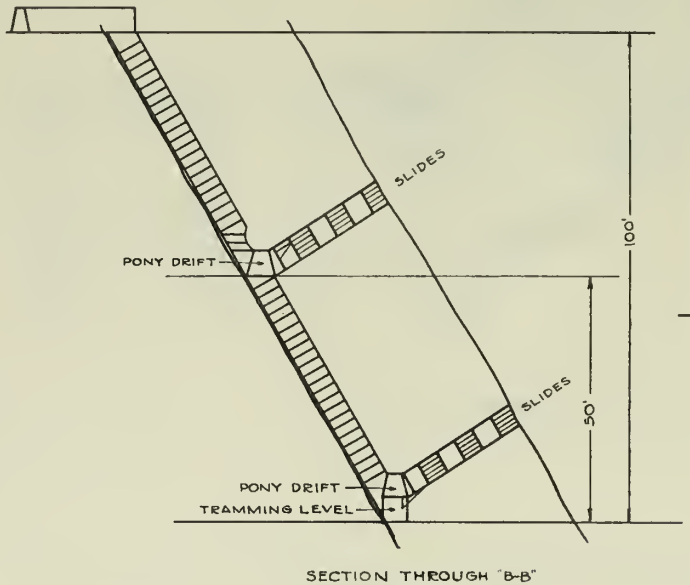
SECTION THROUGH C-C

Showing difference in elevation of cross-cut slides
Draw from central slide may collapse others on side

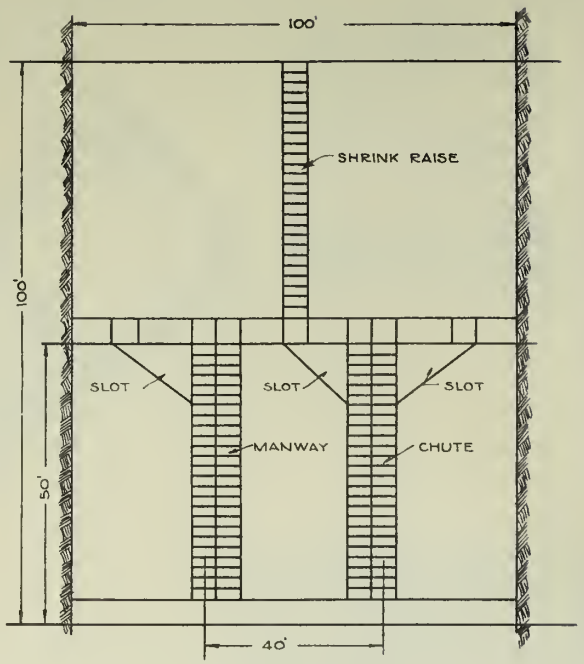
Figure 3.- Shows shrinkage stope on foot wall and position of inclined and cross-cut slides



PLAN - UPPER INSTALLATION



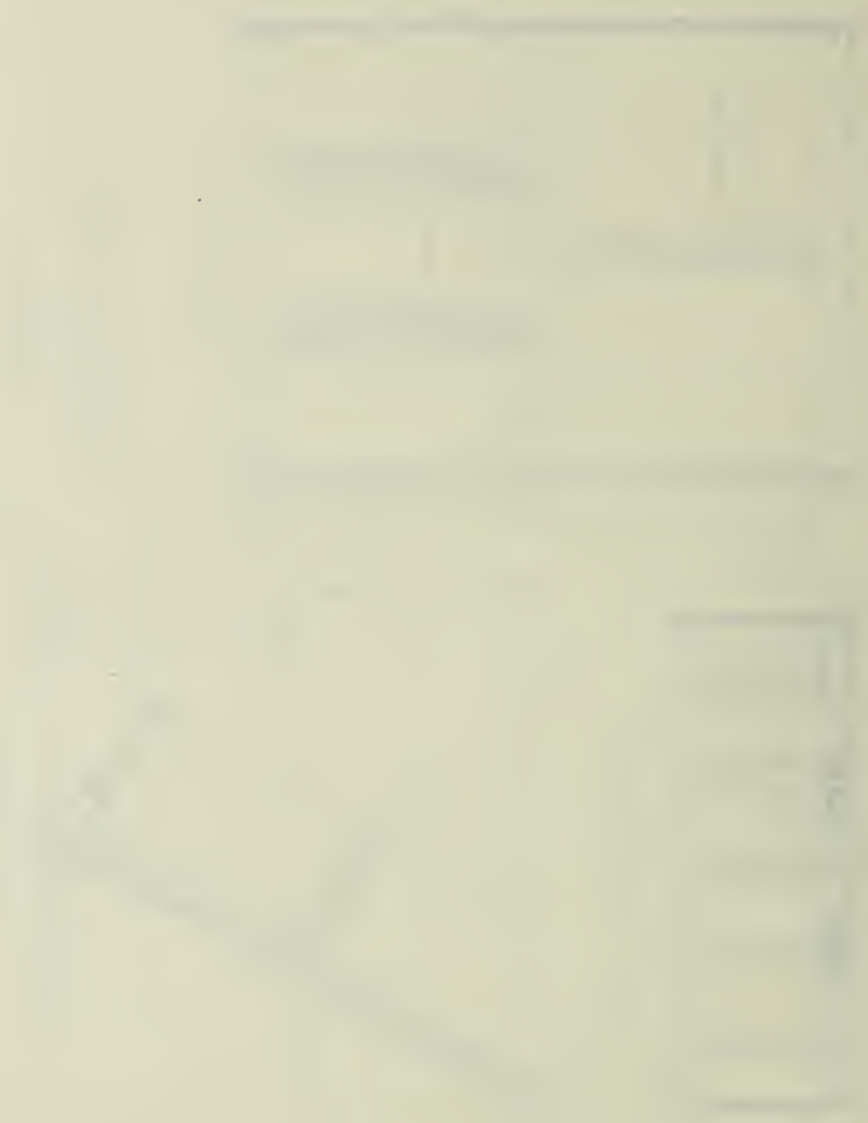
SECTION THROUGH "B-B"

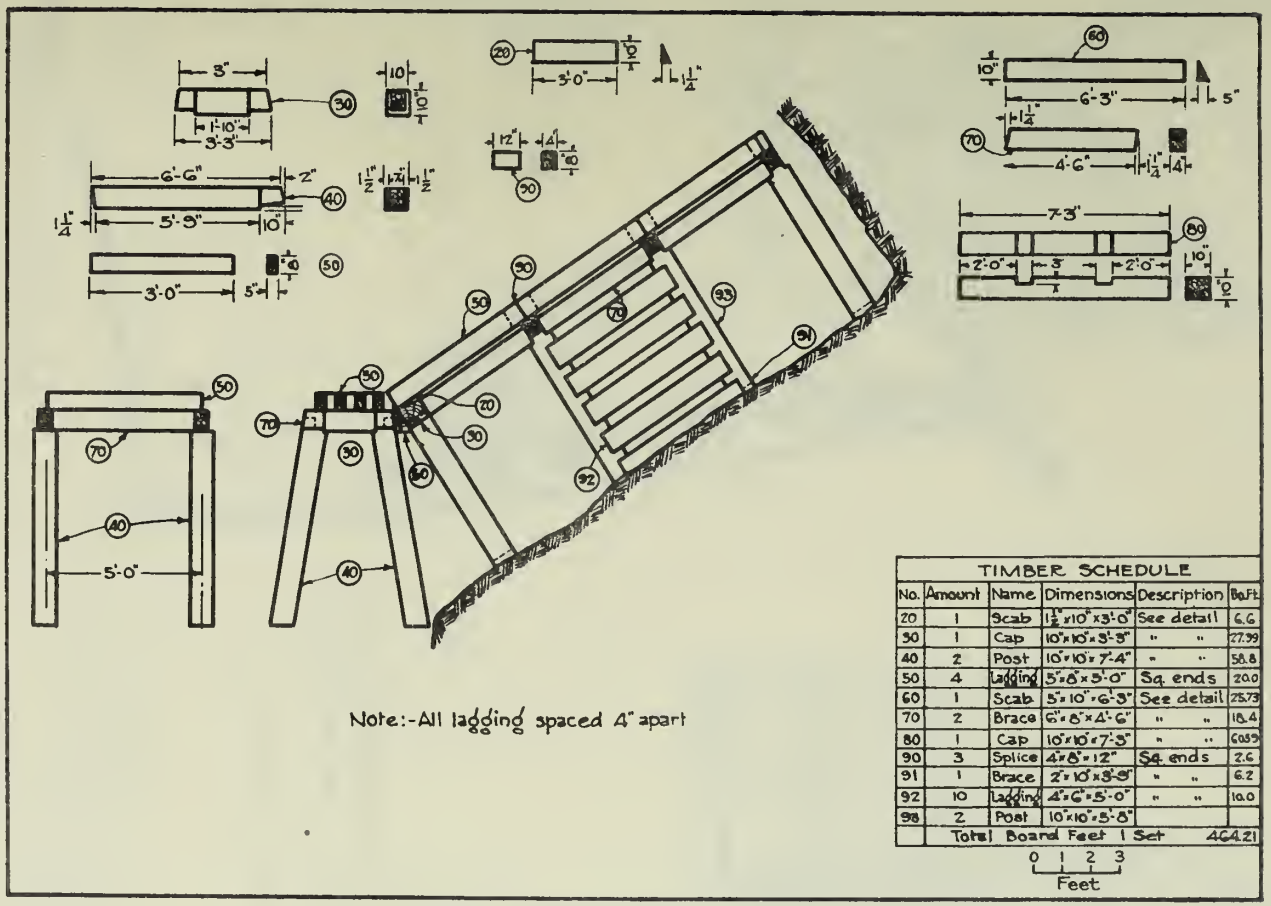


SECTION THROUGH "A-A

TYPICAL CAVING STOPE
OLD DOMINION COMPANY

FIGURE 4





TIMBER SCHEDULE

No.	Amount	Name	Dimensions	Description	Bd.Ft.
20	1	Scab	1½"x10"x3'-0"	See detail	6.6
30	1	Cap	10"x10"x3'-3"	" "	27.99
40	2	Post	10"x10"x7'-4"	" "	58.8
50	4	Lagging	5"x8"x3'-0"	Sq. ends	20.0
60	1	Scab	5"x10"x6'-3"	See detail	25.73
70	2	Brace	6"x8"x4'-6"	" "	18.4
80	1	Cap	10"x10"x7'-3"	" "	6059
90	3	Splice	4"x8"x12"	Sq. ends	2.6
91	1	Brace	2"x10"x8'-8"	" "	6.2
92	10	Lagging	4"x6"x5'-0"	" "	10.0
98	2	Post	10"x10"x5'-8"		
	Total Board Feet 1 Set				464.21

Figure 5.- Timbering of Pony Drift and Slide Old Dominion Co.



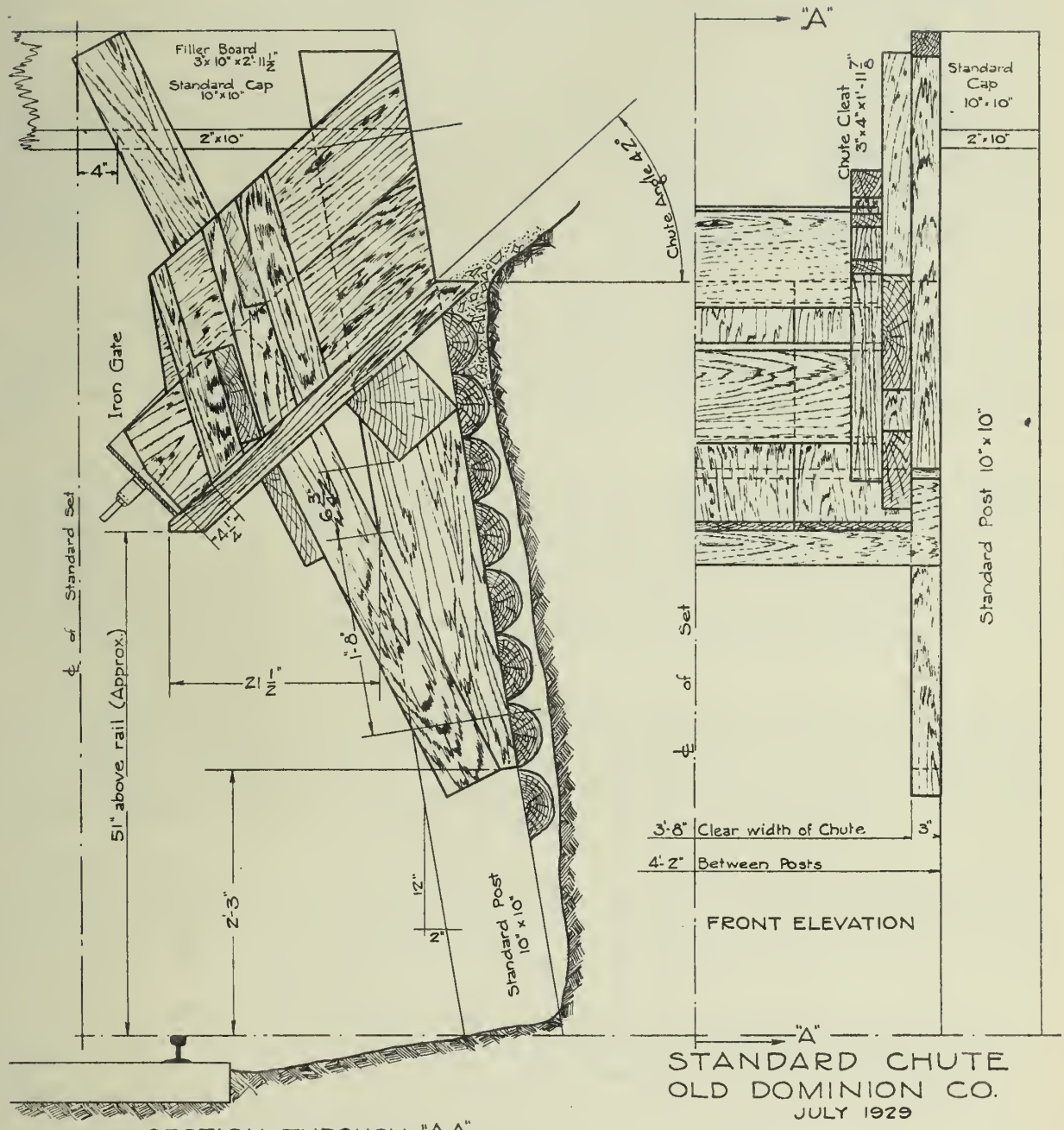

Filler Board 3"x10"x2'-11½"
Standard Cap 10"x10"
2"x10"
4"
Iron Gate
℄ of Standard Set
51" above rail (Approx.)
14½"
6¾"
1'-8"
21½"
2'-3"
12"
2"
Standard Post 10"x10"
SECTION THROUGH "AA"
"A"
Chute Angle 42°
Chute Cleat 3"x4"x1'-11½"
Standard Cap 10"x10"
2"x10"
Standard Post 10"x10"
℄ of Set
3'-8" Clear width of Chute 3"
4'-2" Between Posts
FRONT ELEVATION
"A"
STANDARD CHUTE
OLD DOMINION CO.
JULY 1929
12" 6" 0 1'-0" 2'-0"
SCALE

FIGURE 6

The development and operation of the upper lift is identical with the lower, except that the ore is drawn from the slides into raises instead of into loading chutes. These raises are run midway between two slides. A grizzly set is placed in the raise just below the pony drift and slots are run in either direction to receive the ore from the slides. The grizzlies are made of rails; the interval is 10 inches.

A somewhat different set from that first used is now standard (fig. 5). The short framed cap will usually last the life of the stope, giving a much smaller repair cost than the longer cap. The battering of the posts is intended to throw as much of the weight as possible into the side walls and thus relieve the cribbing and grizzly installation under the pony sets from weight which necessitates repairs. With the exception of the length of the posts, the same timber set is used in both the pony drifts and the slides.

A lagging interval of 4 inches is maintained by using 4 by 6 inch spacing blocks. This is an important detail that must be insisted upon. This arrangement gives access to any point over the slide or pony drift, so that large bowlders which may be throwing weight upon a set during drawing can be drilled and blasted.

After the pony drift has been driven and the slides begun, the shrinkage stope on the footwall can be started. It is not necessary that the cross slides be completed. They need to be driven and timbered only beyond the shrinkage stope limits so that they will not be flooded with broken ore when the time comes for their completion. This is an attractive feature of the system, for the swell from the shrinkage stopes gives considerable production before the development work is completed.

If necessary, controls are placed on the footwall side of the pony sets from which the shrinkage stope can be drawn. If the footwall is flat, regular inclined sets are run into the footwall to get draw points well under the shrinkage ore. Shrinkage stopes are pulled into the transfer raises on the upper lift and into the regular chutes at 20-foot intervals on the lower lift.

Figure 6 shows the standard chute used throughout the mine. Where this chute serves an inclined slide, the chute bottom is kept at least 1-1/2 feet below the bottom of the slide, so that there will be a vertical drop into the chute, and so that short grizzly rails may be put in if necessary.

The width of the shrinkage stope on the footwall is determined by the hardness of the ground. In some places the ore on the footwall is hard quartz. The shrinkage stope is then carried as wide as possible. If the ore is soft, a narrow cut is sufficient.

In timbering the inclines, beginning with the first set beyond the brow set, controls for drawing ore are placed on both sides in every other set. The remaining sets are covered with 4 by 6 inch lagging.

The controls are made by nailing cleats on the posts at 45° to the post and placing a 2 by 12 inch plank between them. The control is placed 18 inches above the bottom of the slide. The upper part of the control set is provided with two 4 by 6 inch lagging, one placed parallel to the collar brace, the other angled across the set about one-third the distance from the top.

After the inclines are run and the shrinkage stope completed, undercutting is begun. The first operation is to cut the draw holes. These consist of untimbered raises beginning at the side of the incline at the draw sets and extending upward at an angle of about 45° for a distance of approximately 9 feet. These raises form an apex midway between the slides. The bottom of the apex between two draw raises from adjoining slides is left higher than the top of the timber of the slides in order that the pillar thus formed may beat part of the weight of the overlying broken ore.

A cut of at least 4 feet is then taken out directly over the length of the inclines.

The next step is to blast another round in the back of the draw holes, and then the pillars between the draw-holes are drilled and blasted. The holes are drilled with stopers and pluggers, using 6-foot steel. From the hanging-wall side of the draw-hole raises, holes are drilled vertically and fanned into the pillar toward the footwall. A pillar from apex to apex between the slides is blasted down as a unit and must always be entirely cut to reduce the probability of a large section of unbroken ground settling on the slides.

The initial cut over the inclines is important. There must be enough room to accommodate the swell from the first slough of ore so that no direct weight is thrown upon the timbers.

A draw hole from the next slide beyond the pillar being shot down should be completed before blasting to give a safe entrance from which to observe the effect of the blast. In many cases it is possible to bar down and drill and blast the back a second time. This would be impossible were a safe entrance not prepared in advance.

As a safety measure in undercutting, no holes may be drilled in a pillar beyond a vertical plane separating two pillars. This is to prevent drilling into missed holes. The undercutting of pillars starts from the footwall side of the vein and progresses to the hanging wall.

The drawing, where the stope is on an inclined vein, must start from the footwall side and be limited to that side for some time in order to get the hanging wall waste to follow on top of the ore. More ore is drawn from the footwall draw points than from those points nearer the hanging wall. As the highest-grade ore is generally on the footwall side of the Old Dominion vein, it is important to recover the footwall ore as completely as possible. The first ore to be drawn is that broken in the shrinkage stope.

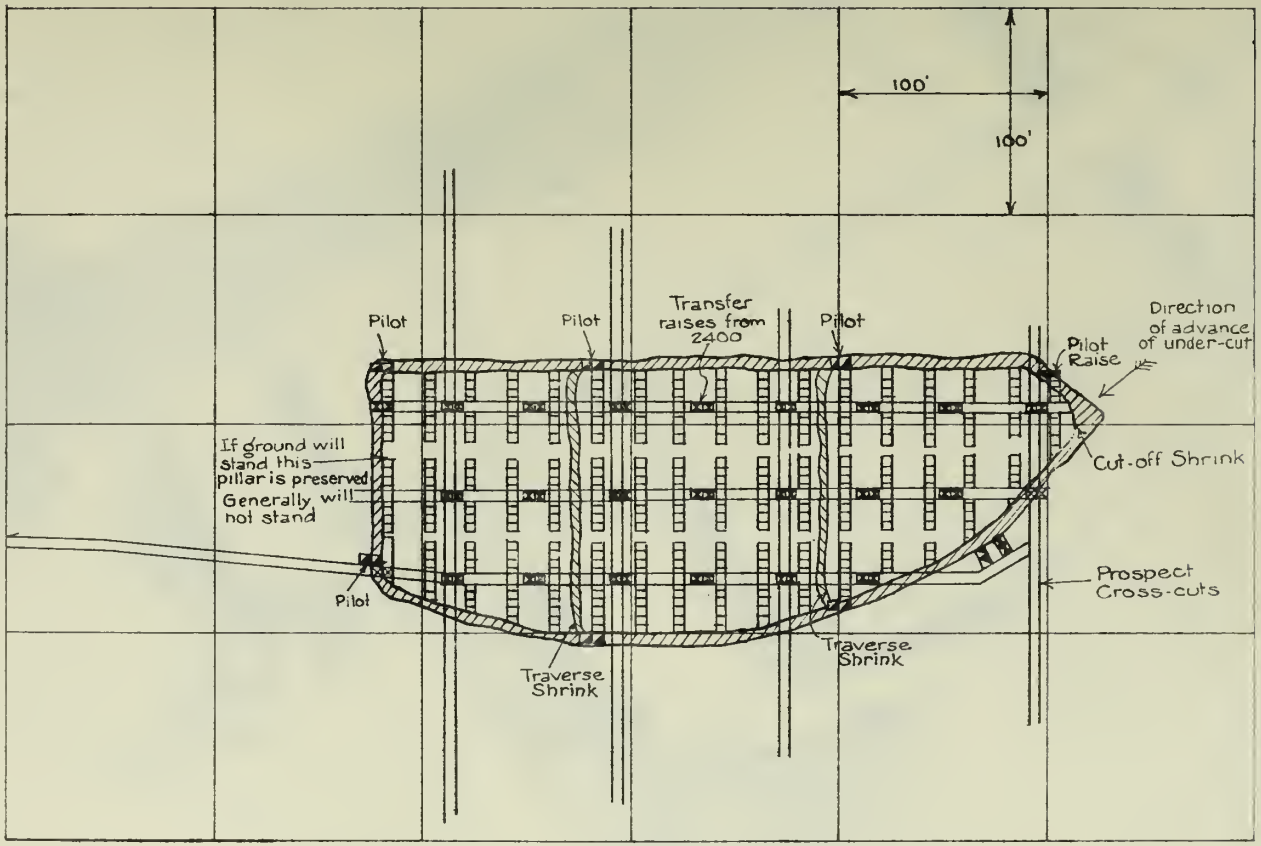


FIGURE 7.- PLAN OF STOPING 2300 LEVEL

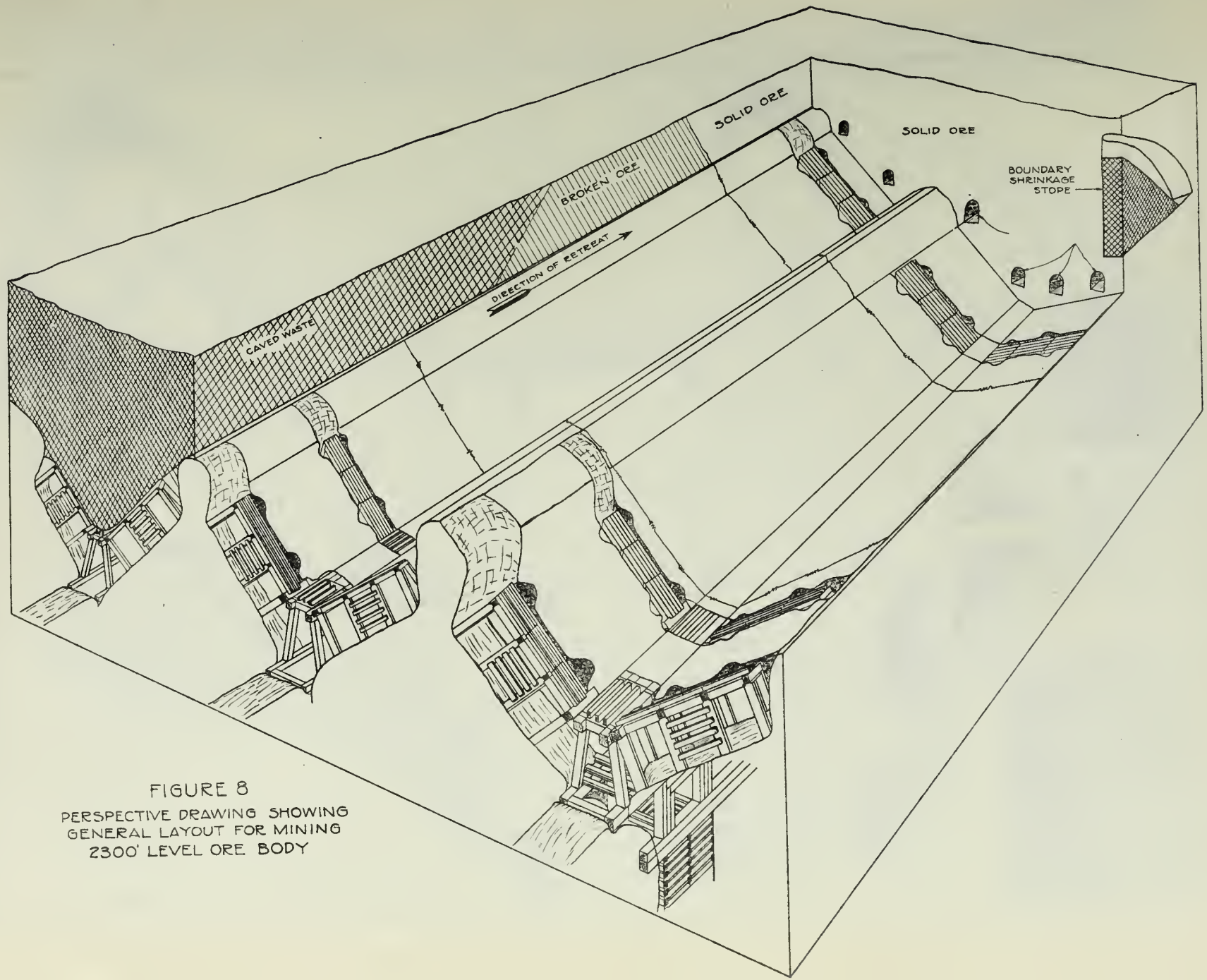
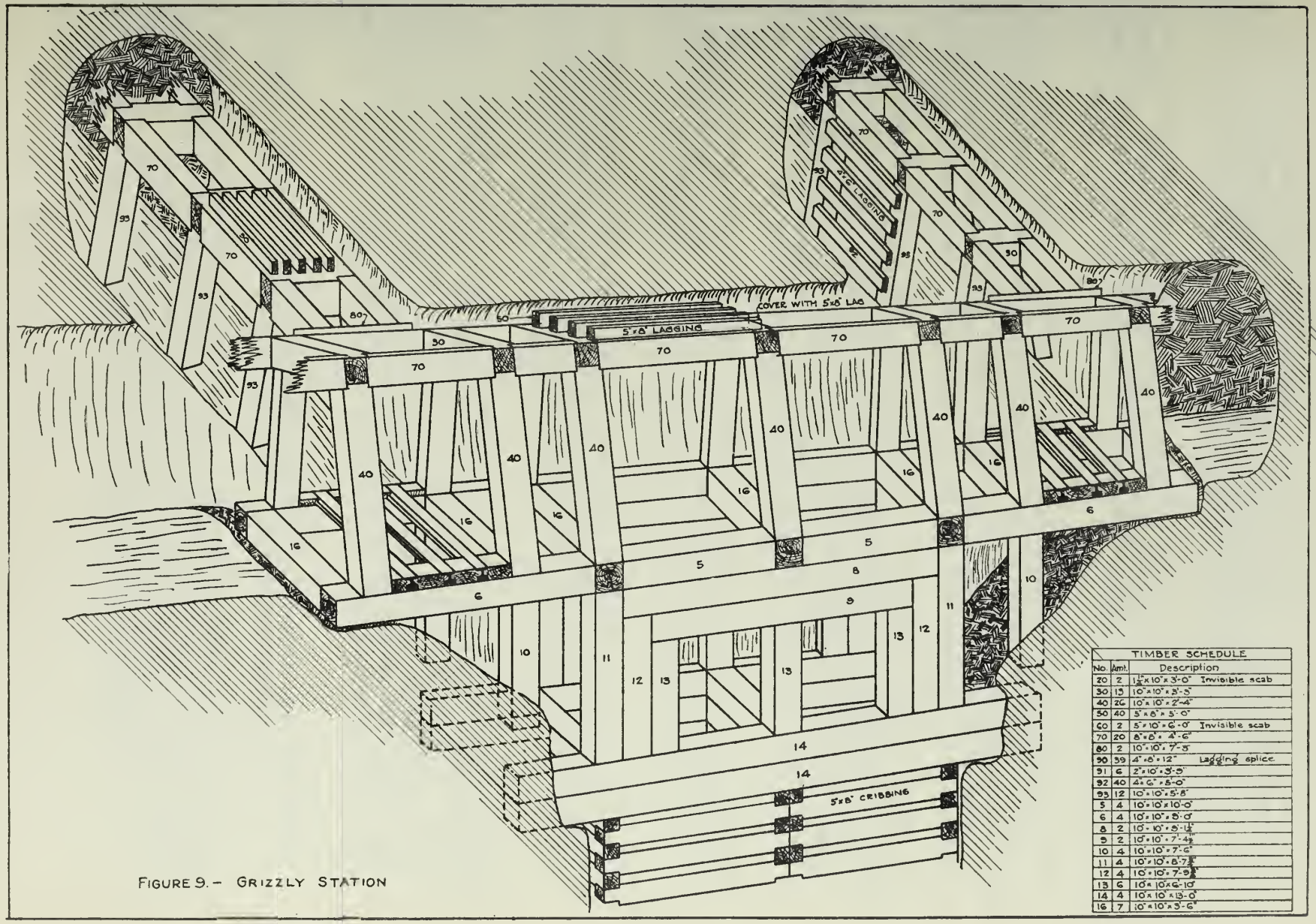


FIGURE 8

PERSPECTIVE DRAWING SHOWING
GENERAL LAYOUT FOR MINING
2300' LEVEL ORE BODY



TIMBER SCHEDULE		
No.	Amt.	Description
20	2	1½"x10"x3'-0" Invisible scab
30	13	10"x10"x3'-5"
40	26	10"x10"x2'-4"
50	40	5"x8"x5'-0"
60	2	5"x10"x6'-0" Invisible scab
70	20	8"x8"x 4'-6"
80	2	10"x10"x 7'-8"
90	39	4"x8"x12" Lagging splice
91	6	2"x10"x3'-5"
92	40	4"x6"x8'-0"
93	12	10"x10"x5'-8"
5	4	10"x10"x10'-0"
6	4	10"x10"x8'-0"
8	2	10"x10"x9'-1½"
9	2	10"x10"x7'-4½"
10	4	10"x10"x7'-6"
11	4	10"x10"x8'-7½"
12	4	10"x10"x7'-9½"
13	6	10"x10"x6'-10"
14	4	10"x10"x13'-0"
16	7	10"x10"x3'-6"

FIGURE 9.— GRIZZLY STATION

A fairly large ore body, having a maximum cross section of 330 by 200 feet and a thickness of 100 feet, is being worked on the 2,300 level west by means of the Morenci inclined slide system. An ore body of this size and shape requires different treatment from a small inclined ore body in the vein.

As production from this tope has just started, it is impossible to give final results. The procedure may have to be varied, for it is nearly impossible to foresee every difficulty that may arise in an entirely new area. Figure 7 shows the general plan.

The ore body will be cut off on all sides by a shrinkage stope. Transverse shrinkage stopes are run to prevent the weight of an overthrust from settling on advanced development work and also to prevent the arching of the back over a large area when the section is drawn. The transverse stope needs to be run only about 40 feet high. The height is left to the judgment of the operator.

Each transfer raise from the 2,400 level serves four slides. These four slides furnish the draw points for a 40 by 40 foot section. The entire ore body on the 2,300 level will be considered as one stope. That is, the relation of advance development and undercutting to drawing will be such that the boundary between broken ore and waste will be a plane dipping about 50° in the opposite direction from that in which the stope is advancing.

The boundary shrinkage stope around the block is only partly completed before undercutting starts. However, it is completed to the top of the ore in the section to be undercut before that portion of the stope is cased. The advance work in this shrinkage is left on an incline.

The transverse shrinkage stope is not run until after completion of the outside stope beyond the position of the transverse stope and after the undercutting on the north side has reached the transverse line. This is to prevent the settling of a large block en masse.

The undercutting procedure is very similar to that previously described. The direction of advance of the undercut is shown in Figure 7. Wherever possible in undercutting, at least two rounds will be blasted above the timbers. Great thoroughness in undercutting appears to be amply justified. Although the cost of the undercut work is increased, complete undercutting tends to prevent large unbroken sections from being left in a stope; also, fewer large bowlders occur in the ore.

Figure 8 is a perspective drawing showing the general plan for mining this ore body. Figure 9 shows the details of a grizzly installation.

An estimated stoping cost, from the the grizzly installation up, for mining this ore body is given in Table 1. The powder, repairs, and drawing costs are estimated. These figures are based upon the costs in a unit block (fig. 7) 40 by 40 feet in cross section and 100 feet high, which contains 12,800 tons. The cost details for each operation in a block are shown in Table 2.

Table 1. - Estimated Direct Mining Cost of 2,300 Level Ore Body by Morenci Inclined-Slide Method

	Per ton
Development	
Labor on (slides, pony drift, grizzly inst.)	$0.060
Timber (slides, pony drift, grizzly inst.)	0.038
Shrinkage	0.021
Undercutting	0.136
Drawing at 40 tons per man-shift	0.130
Powder (for development and bulldozing)	0.040
Repairs	0.050
	$0.475

Table 2. - Cost Details

		Final direct mining cost per ton ore
40-foot pony drift at $5.75	$230.00	
80-foot slides at $4.25	340.00	
Grizzly, 33.6 shifts at $5.69 each	191.18	
	761.18	$0.060
Undercut		
Cut-over inclines, 80 feet at $0.50	40.00	
16 draw holes at $4.00 each	64.00	
40 feet over pony drift at $4.75	190.00	
Undercut on line of draw holes, 160 feet at $3.25	520.00	
4 pillars at $30 each	120.00	
	934.00	.136
Shrinking, 1,007 yards at 60 cents	604.20	.021
Timber		
Slides, 60 feet at 70 board feet per linear foot	147.00	
Pony drift, 40 feet at 67 board feet per linear foot	93.60	
Grizzly installation, 2,815 board feet total	98.52	
	339.12	.058
Drawing at 40 tons per man-shift		.130
Powder (development and bulldozing)		.040
Repairs		.050
Total		$0.475

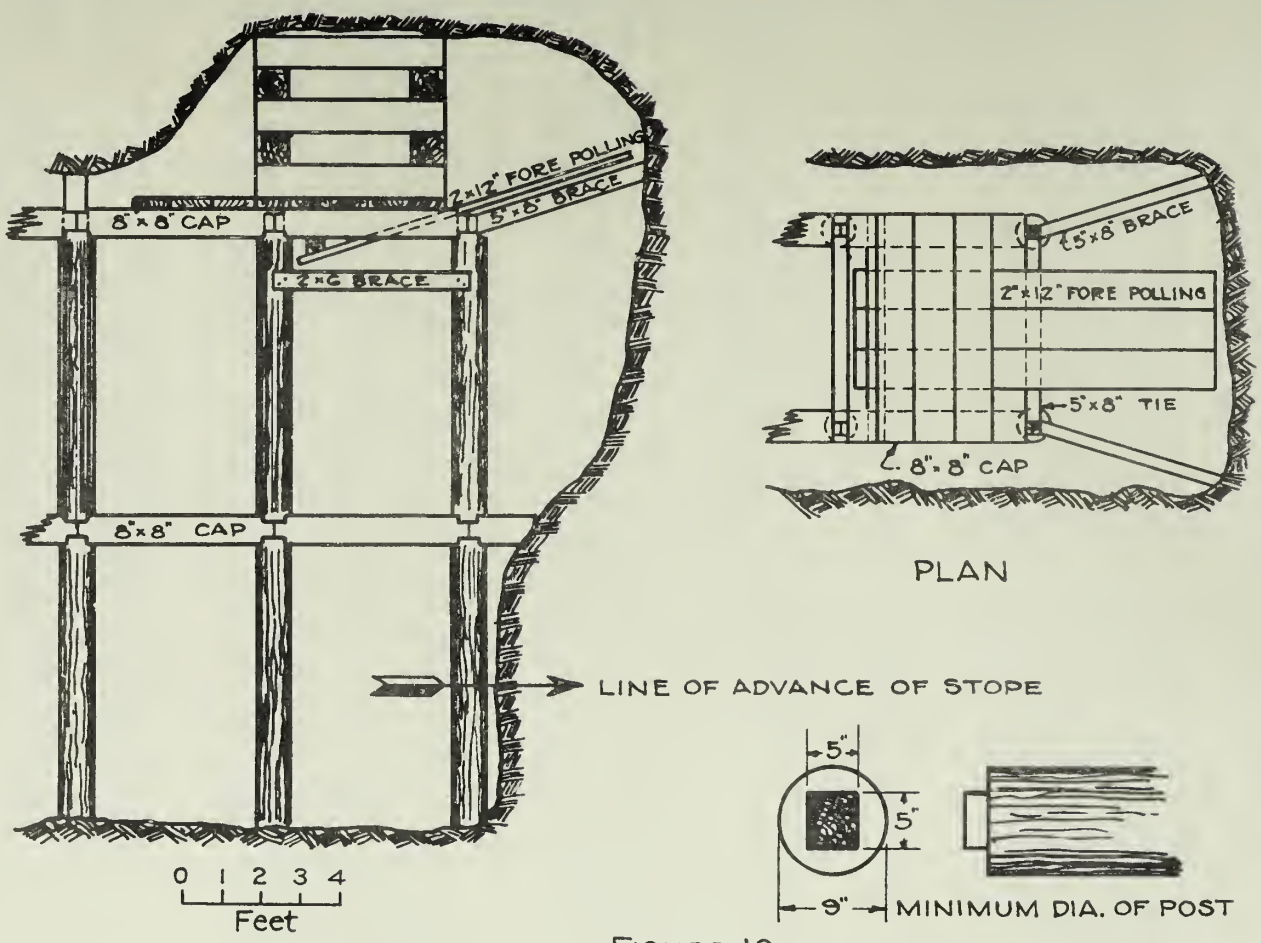


FIGURE 10
METHOD OF TIMBERING SQUARE SET STOPES



Square-Setting.--Square-setting is largely confined to areas that must be supported after mining to minimize the danger of breaking a dacite flow and permitting the entrance of water from the overlying Gila conglomerate.

The standard practice is to run 15-foot sections across the width of the vein. The opening of more ground than this at one time is unsafe in nearly all the square-set areas.

In prospecting ground that is to be mined by the square-set method, intervals that are multiples of 30 feet are used in running the raises up on the footwall, so that they will fit into a stoping scheme that allows one raise for every two 15-foot sections. The raises have two square compartments each 3 feet 4 inches in the clear; one is used as a manway and the other for an ore chute. One compartment of the raise is kept in each section, and the manway is in the section to be taken out last. Waste is brought into the stope through the compartment used for ore.

Where the stope is over eight sets wide, a raise may be run into the hanging wall for filling on the hanging-wall side of the stope.

The broken ore is generally shoveled or wheeled into the chutes. Sometimes in wide stopes slides are used or another chute may be carried up toward the hanging wall by keeping a set open and lined off.

Waste filling is obtained from development work or waste raises. Slides are used for deflecting the waste toward the hanging wall. A part of the waste is placed by means of wheelbarrows or by hand-shoveling. The wheelbarrows are filled through a temporary chute built in square-sets of the floor above.

Usually not over two floors are taken out before filling.

Eight by eight inch posts and caps and 5 by 8 inch ties are used for square-sets except in excessively heavy ground, where 10 by 10 inch timber is used. The caps are placed parallel to the line of advance of the stope, or at right angles to the strike of the vein. As this permits the set to be lagged from cap to cap and head boards to be placed over the tie to protect the miners while advancing, the necessity of a bridge is eliminated (fig. 10). Wherever spiling ground is encountered this order is reversed so that the spilling may be driven over the cap.

In most cases filling is done from the footwall side. This requires that the footwall side be kept two or more sets higher than the hanging-wall side.

Shrinkage.--Shrinkage stoping is at present used in only one vein which is relatively narrow, having a maximum width of 25 feet. A drift in the vein is timbered with standard tunnel sets and chutes put in on one or both sides of the drift, depending upon the thickness of the vein. Chutes are placed at not over 15-foot intervals. Only the width of the drift is broken on the sill floor.

Generally a pilot raise is driven to the level above before stoping begins for ventilation and for bringing in supplies. After the stope reaches a point midway between the two levels, supplies are brought in from above. Compressed air is brought in through the raise by means of a 1-1/2-inch hose with a manifold on the end. Single-compartment cribbed raises are carried up through the broken ore on either end of the stope. Unless speed is necessary, mining and drawing are done on separate shifts. However where bornite is an important mineral constituent, the stope must be run as rapidly as possible, in order to prevent excessive oxidation and subsequent losses of copper in milling.

Inclined Top-Slicing.--Top-slicing is used where the grade of the ore is so high that clean mining is necessary, and where it is permissible to cave the overlying ground.

Inclined top-slicing has been combined with flat top-slicing very successfully in ore bodies 10 to 80 feet wide. A flat section 20 feet wide is run across the ore body from footwall to hanging wall. From this flat slice inclined slices are run parallel to the strike of the vein. The flat section is in no way essential to the success of inclined slicing; it was introduced as a safety measure to provide two exits to each working place in case of a cave-in. Projected to the horizontal, the length of the inclined sections is 35 feet. One complete stoping section, then, extends 70 feet along the strike of the vein. An 11-foot stull is used in the flat section; this permits the use of a 9-foot stull in the inclined section.

Two-compartment cribbed raises, 10 feet apart in the clear, are run on both sides of the flat slice before mining starts. A manway is maintained in each raise. From these raises slots at right angles to the line of the inclined sections are cut out in advance of each slice. The slots are run as narrow as it is possible to cut them. The bottom slope is 40°. This permits ore to flow down the inclines into the slot and then into the raise.

If the ore body is narrow and requires only one raise on each side of the flat section, the raises must be run on the footwall; otherwise, the length of the slot would be excessive.

Round timber is used for stulls and stringers and 2-inch by 12-inch by 10-foot boards are used for mat. The stulls are notched on one end, so that they will remain in place when put against a round stringer. This notch must not be too deep or the stull may split. Stulls are drilled for blasting in the framing shed on the surface. The stringers in the inclined section are laid at right angles to the line of advance--from foot to hanging. In the flat section over the slots they are laid in the opposite direction. In the remainder of the flat section between slots they are laid as in the inclinded sections. Each line of mat overlaps about 8 inches. An occasional line of 2 by 12 inch lagging is placed behind the stulls to break the fall of material rolling down the incline. Mat is laid as the panel advances, so that the ore has a floor upon which to slide. The lagging is nailed to the stringers.

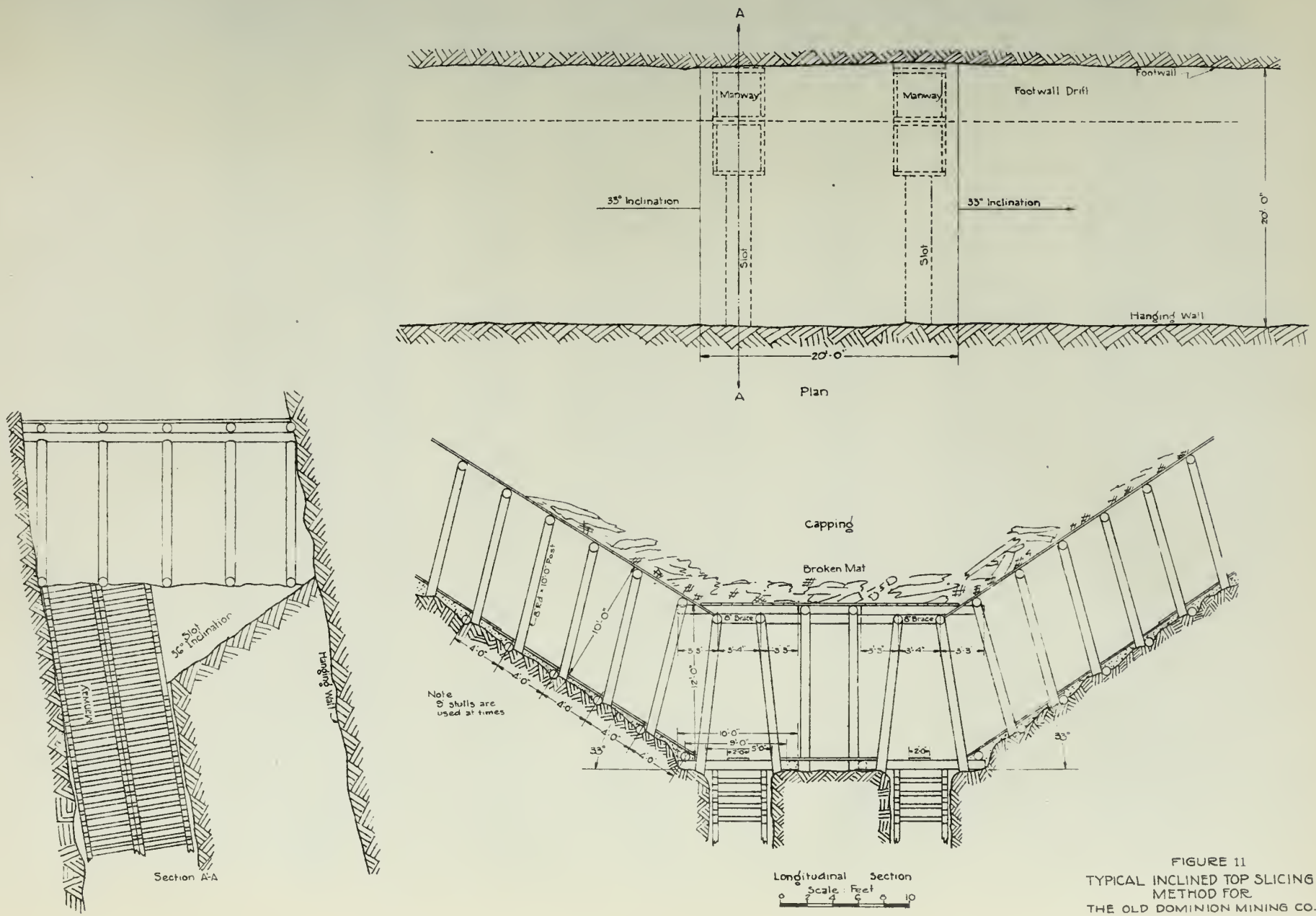


FIGURE 11
TYPICAL INCLINED TOP SLICING METHOD FOR THE OLD DOMINION MINING CO.

Figure 11 shows an idealized plan and section of an inclined top-slice and the timbering details.

Unless the ground is very heavy, the section is blasted down as a unit. If, however, one part of the stope is very heavy, a single panel may be shot down. At least two panels should be completed before the first one is blasted, for otherwise there would be danger of getting waste into the ore through the broken mat.

It is essential that the timber on apexes be joined, so that there will be no possibility of waste coming through at this point. These apexes give no trouble if thorough timbering is done. If an unbroken pillar should be left on the apex, it will give considerable trouble on the floor below. All explosives used for blasting timber in bringing down top-slices are the permissible type used in coal mining.

When an extensive length of vein is to be mined by this method, mining starts on the end of the ore body farthest from the shaft. Several sections may be mined at the same time, but the farthest is always kept one floor ahead of the next section.

Open-Stope Mining.--Several narrow flat veins are mined by running raises up the dip on the footwall at 25-foot centers to the top of the ore, then mining the pillars down. Where timbering is necessary in the raises, standard Morenci slide sets are used. The raise sets are reclaimed. Stulls are placed in the stope as necessary. If the back is sufficiently good, no timber is used, except that an occasional row of stulls is put in and 2 by 12 inch lagging is placed behind the stulls. This line is to protect the men working below in the stope. Where the dip is less than 35°, steel sheets are placed on the bottom of the raises to facilitate the flow of ore.

In mining the pillars, inclined cuts 10 feet wide at about 45° to the line of the raise are taken. No shoveling from a flat bottom needs to be done. Cuts from adjacent raises apex in the middle of the pillar. Drilling is done with pluggers.

Miscellaneous use of the Morenci Slide.--The Morenci timbered slide has been used to excellent advantage in shrinkage operations where an ore shoot bottoms between two mining levels. Raises are usually run at 60-foot intervals from the haulage drift to the bottom of the ore, or a little below the bottom. Slides are then run on the vein, apexing midway between raises. A regular shrinkage operation is then carried on above the slide timbers. An intermediate tram is thus avoided. This method requires either the loss of some ore or the mining of some waste. The grade of the ore is the influencing factor in making the choice.

UNDERGROUND TRANSPORTATION

In addition to the 1,500 tons of ore, about 400 tons of waste is hoisted to the surface daily. Separate storage pockets for ore and waste are provided at each haulage level. On some levels where both direct-smelting and milling ore are handled, pockets are provided for both.

The skip capacity is 3-1/2 tons. The average length of hoist is 1,700 feet. All hoisting is done on two shifts six days a week, leaving an extra shift and Sundays for inspection and necessary repairs in the shaft.

The ore is handled on the levels in 20 cubic-foot side-dump cars. A full train consists of 35 cars. Four-ton trolley electric locomotives make the trips to the shaft. Three-ton locomotives are used for gathering. These engines are operated on a 250-volt direct current.

Four-ton storage-battery motors are used on the lowest levels when being opened up and before it is feasible to introduce trolley haulage. Large quantities of water are always encountered in opening up a new level. No type of motor haulage is practicable until the level is drained sufficiently to confine the water to a ditch. Trolley locomotives are not used until excessive drip from the back has stopped.

The maximum haul is 6,000 feet. The average is 3,100 feet. The usual mine grade is 1/2 per cent, in favor of the loaded trains. Track gage is 18 inches. Thirty-pound rails are used for motor haulage. On intermediate levels 12-pound rails are used, unless a large amount of hand-tramming is to be done. In that case, 24-pound rails are installed. For motor haulage, 40 feet is the minimum radius for curves. For hand-tramming, radii are as short as 15 feet.

PUMPING

Pumping is a major operation. During 1928 a daily average of 3,659,000 gallons was pumped. Approximately two-thirds of this water originates between the two bottom levels (2,400 and 2,600).

Water is pumped from the 2,600 to the 2,200 level, then to the 1,200 level, and from there to the surface or to the drain tunnel 40 feet above the 500 level.

The 2,600-level pumping equipment consists of two 1,600-gallon capacity two-stage and one 500-gallon two-stage centrifugal pumps.

The 2,200-level installation consists of two 1,000-gallon capacity quintuplex electric pumps and one 1,000-gallon capacity centrifugal pump, all under a head of 1,000 feet.

The 1,200-level installation consists of four 4-cylinder triple-expansion steam pumps (21 gallons per revolution, average speed 52 revolutions per minute for each pump) under a 1,000-foot head; one quintuplex electric pump of 1,000-gallon capacity at 1,000-foot head, and one 1,500-gallon five-stage centrifugal pump under a 650-foot head.

The temperature of the water from the lower levels is 109°F. With the exception of a small amount of water that seeps through old oxidized country, the mine water is not acid. A small underground precipitating plant removes the copper and neutralizes the acid water before it reaches the sump.

CONTRACT AND BONUS SYSTEMS

At present, 36 per cent of all labor in the mining department is paid on a contract or bonus system. This is exclusive of electricians, pumpmen, hoistmen, nippers, and supervisors. The straight contract is employed much more than the bonus plan.

The contract rates and the bonus standards are set by the bonus department, subject to the approval of the general mine foreman.

It is impracticable to maintain a uniform rate throughout the mine under either plan because of the wide variations in heat and hardness of ground. Nine different classes of ground are necessary to make the earnings of the men even approximately uniform in different parts of the mine. Heat imposes several other variations of rates.

Generally, all men on one contract are paid according to the same wage scale and share equally in the earnings in excess of that wage.

Occasionally men are classed as muckers, miners, or timbermen, and paid the corresponding wage. All earnings over the wages in that case are equally divided; the division is not made proportional to the workers' wage scale.

Drifting, raising, and nearly all stoping, including undercutting, are on contract. Sometimes stopes in very heavy ground are not placed on contract. If such stopes are placed on contract, the tendency is to work too rapidly in order to beat the standard and thus lose the stope.

Track-laying is sometimes placed on contract or bonus. All tramming and haulage are done on the basis of a day's pay. The bonus plan is used in part of the stopes. In a bonus system, an allowance in man-shifts is made for any piece of work based upon the average time required to complete the job on day's pay. If this work is completed in less time than required by that standard, an allowance of 50 per cent of the difference is given the men. The standards in the bonus system are placed about 25 per cent higher than in the contract system, which permits the average miner to make approximately the same money as if on contract.

The drill sharpening in the mine blacksmith shop is on bonus. The shop foreman does not participate in this bonus.

No production bonus is paid to the foremen or shift bosses.

FIRE-FIGHTING AND RESCUE WORK

The operating companies of the district maintain a central station (Globe-Miami Mine-Rescue Association) with a director, attendants, and adequate equipment to care for all emergencies. A crew of 15 trained men is maintained at the mine. Their position in the mine is always known, so that they may be readily called for duty. These men are paid a small bonus.

The Globe-Miami Mine-Rescue Association is also in charge of all first-aid training. Contests, with prizes to winning teams, are held each year. Safety work at Old Dominion is given the status of a major operation. No method of working is permitted which involves any extra hazard, regardless of the saving that might be effected by using the more dangerous method. A safety inspector is in charge of all safety work. Every month a workmen's committee, with the inspector, foreman, and shift boss, visits and grades every working place. The membership of the committee is changed frequently, so that eventually every miner will have served.

Periodical underground safety meetings are held on every run. Any proposals made at these meetings for increasing safety are answered by the general foreman. If these proposals are not put into practice, an explanation is given showing that the plan would not be feasible.

In addition to these meetings, the foremen and shift bosses have a regular monthly conference.

VENTILATION

The active mining zone is in high-temperature rock, and the principal ventilation requirement is the improvement of comfort conditions by lowering the temperature and providing air motion. The mine is mechanically ventilated by three large surface fans, operating as blowers on the C, Kingdon, and Grey shafts. These fans are aided by large natural-draft pressures and underground booster fans. The interconnected A and Pump shafts form a common upcast for the C and Kingdon shaft downcast circuits, and the upper part of one compartment of the Kingdon shaft is connected to a line of upcast raises that serve as the return for the Grey shaft downcast circuit.

The main fans are multibladed, forward curved blade, centrifugals. The two on the C and Grey shafts have housings for reversing the direction of flow. A recent survey showed a total intake of approximately 229,000 cubic feet of surface air per minute--100,000 through C shaft, 69,000 through Kingdon shaft, and 60,000 through Grey shaft--equivalent to about 760 cubic feet per minute per man employed underground on the day shift.

Most of the stope ventilation is by ascending currents, although occasionally downcast currents are used. A large number of doors and a few small booster fans are used to control the direction of flow. Natural drafts are large and maximum pressures have been calculated as ranging from 1.5 to 3.0 inches water gage between the various downcast and upcast circuits to the 2,000 level. Virgin rock temperatures are high--about 109° on the 2,600 level, the lowest development level. All working places off the major air circuits are provided with auxiliary ventilation through fan-tubing units using small high-speed centrifugal fans and 12 and 16 inch diameter tubing. In the lowest development workings 22-inch diameter steel pipe, made from oil drums, is used on vertical lines in shaft compartments, and some exceptionally large fans are required. Radial-blade fans predominate in auxiliary fan service, although a number of designs with forward-curved blades are also used.

Air samples show that the quality of the mine air, as to chemical composition, is good throughout the mine.

Table 3. - Summary of costs in units of labor, power, and supplies

Name of mine: Old Dominion mine. Period covered: Year of 1928.

Tons of ore mined and hoisted: 350,661.

	Development	Mining	Total
A. Labor (man hours per ton):			
Breaking) Timbering)	.738	0.637	1.38
Mucking)			
Haulage and hoisting			0.95
Supervision			0.24
General (all other underground).			1.16
Total labor underground			3.73
Labor, percentage of total cost.			60.8
Surface shifts chargeable to underground mining			0.51
B. Power and supplies:			
Explosives (lbs.)	7.3 (per ft.) 40% gelatin	0.28 (per ton) 40% gelatin	
Timber (bd. ft.)	60 per ft.	4.8 bd. ft. per ton	
Power (per ton):			
(1) Air compression			22.8 hp.hrs.
(2) Hoisting			13.1 do.
(3) Pumping			34.2 do.
(4) Ventilation			6.6 kw.hrs.
(5) Lighting			0.7 do.
(6) Haulage			2.0 do.
Supplies, exclusive of timber and powder, percentage of total supplies and power			26.6 per cent
Supplies and power, percentage of total cost			34.6 per cent
C. Percentage of total cost	26.3	73.7	100.0

Table 4. - Development costs in units of labor and supplies, 1928

	Size	Feet per man-shift	Pounds powder per foot	Board feet timber per foot
Timbering drifts, hand tram	8 by 6	- -	- -	60
Timbering drifts, motor	9 by 7	- -	- -	80
Average level drift		0.83	9.19	60
Intermediate drifting	6 by 4	0.89	5.72	
Raising	5 by 9	1.22	6.01	120
Intermediate raising		1.13	5.08	
Sinking	17 by 6	0.35		251
Grand average	- -	0.88	7.30	60

Table 5. - Tons mined per unit of labor and supplies

Year	Tons mined	Tons per man-shift			Pounds powder per ton	Board feet per ton
1928	350,661	1/ 9.27	2/ 1.98	3/ 1.39	0.28	4.76

1/ Stoping shifts only.
2/ All mine shifts and surface shifts chargeable to mine.
3/ Total surface and underground shifts (including concentrator).

Table 6.- Tons per man-shift mined by each mining method

Mining method	Tons per man-shift
Square-set	5.7
Inclined top slice	9.5
Morenci slide cave	17.3
Shrinkage	15.0
Average	9.27

TABLE 1.

CURVE DATA FOR MOTOR DRIFTS - 4 FEET 0 INCHES
STANDARD CAP.

Radius feet	Number of sets 1	2	3	4	5	6	7	8	9	10	11	12	13	Collar braces Long	Short	Diff.	Cut	Diagonal	Angle X
12-1/2'	19°18' 2.12'	38°36' 4.38'	57°54' 6.92'	77°12' 9.98'	96°30' 14.01'									4'2"	2'6-3/4"	19-1/4"	13/16"	5'2-3/16"	19°18'
15'	16°30' 2.18'	33°00' 4.44'	49°30' 6.92'	66°00' 9.74'	82°30' 13.16'	98°00' 17.26'								4'2"	2'8-7/8"	17-1/8"	11/16"	5'3"	16°30'
17-1/2'	14°26' 2.22'	28°52' 4.50'	43°18' 6.95'	57°44' 9.65'	72°10' 12.75'	86°36' 16.49'	101°02' 21.24'							4'2"	2'11-1/3"	14-7/8"	1/2"	5'4"	14°26'
20'	12°48' 2.24'	25°36' 4.54'	38°24' 6.96'	51°12' 9.58'	64°00' 12.50'	76°48' 15.85'	89°36' 19.86'	102°24' 24.87'						4'2"	3'1-1/8"	12-7/8"	7/16"	5'4-5/8"	12°48'
22-1/2'	11°32' 2.27'	23°04' 4.59'	34°36' 7.01'	46°08' 9.58'	57°40' 12.39'	69°12' 15.52'	80°44' 19.13'	92°16' 23.41'	103°48' 28.69'					4'2"	3'2-7/16"	11-9/16"	3/8"	5'5-1/8"	11°32'
25'	10°28' 2.29'	20°56' 4.62'	31°34' 7.03'	41°52' 9.56'	52°20' 12.28'	62°48' 15.26'	73°16' 18.59'	83°44' 22.41'	94°12' 26.90'	104°40' 32.39'				4'2"	3'3-1/2"	10-1/2"	3/8"-	5'5-1/2"	10°28'
30'	8°50' 2.32'	17°40' 4.66'	26°30' 7.07'	35°20' 9.56'	44°10' 12.17'	53°00' 14.96'	61°50' 17.97'	70°40' 21.27'	79°30' 24.95'	88°20' 29.14'	97°10' 34.01	106°00' 39.81'		4'2"	3'5-1/8"	8-7/8"	5/16"	5'5-3/4"	8°50'
40'	6°46' 2.36'	13°32' 4.75'	20°18' 7.16'	27°04' 9.63'	33°50' 12.16'	40°36' 14.80'	47°22' 17.54'	54°08' 20.88'	60°54' 23.52'	67°40' 26.81'	74°26' 30.38'	81°12' 34.28'	87°58' 38.60'	4'2"	3'7-1/16"	6-15/16"	1/4"-	5'6-3/4"	6°46'
50'	5°28' 2.39'	10°56' 4.79'	16°24' 7.21'	21°52' 9.66'	27°20' 12.16'	32°48' 14.72'	38°16' 17.35'	43°44' 20.06'	49°12' 22.89'	54°40' 25.85'	60°08' 28.95'	65°36' 32.23'	71°04' 35.71'	4'2"	3'8-15/16"	5-1/16"	3/16"	5'7-7/16"	5°28'
60'	4°36' 2.41'	9°12' 4.83'	13°48' 7.26'	18°24' 9.72'	23°00' 12.21'	27°36' 14.70'	32°12' 17.32'	36°48' 19.96'	41°24' 22.67'	46°00' 25.47'	50°36' 28.36'	55°12' 31.37'	59°48' 34.50'	4'2"	3'9-3/8"	4-5/8"	1/8"+	5'7-5/8"	4°36'
75'	3°42' 2.42'	7°24' 4.85'	11°06' 7.29'	14°48' 9.74'	18°30' 12.24'	22°12' 14.71'	25°54' 17.25'	29°36' 19.82'	33°18' 22.43'	37°00' 25.10'	40°42' 27.82'	44°24' 30.61'	48°06' 33.48'	4'2"	3'10-5/16"	3-11/16"	1/8"	5'8-1/16"	3°42'
100'	2°48' 2.44'	5°36' 4.89'	8°24' 7.34'	11°12' 9.81'	14°00' 12.23'	16°48' 14.77'	19°36' 17.27'	22°24' 19.80'	25°12' 22.35'	28°00' 24.93'	30°48' 27.54'	33°36' 30.19'	36°24' 32.88'	4'2"	3'11-1/8"	2-7/8"	1/8"-	5'8-1/4"	2°48'

Upper figures are total curvatures; lower figures are tangent distances, in feet.

"Diagonal" means diagonal distance between posts of adjacent sets, measured at bottom of cap.

"Angle X" is the angle turned per set.

The standard cap is 4 feet, 0 inches in the clear between posts; bottom of cap is 7 feet, 6 inches above rail.

Collar braces are 4 inches by 6 inches, lengths indicated being measured on center line.

Collar braces are centered on the posts; that is, 4 feet 10 inches center-to-center across drift.

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MINING METHODS AT THE MORNING MINE OF THE FEDERAL MINING AND SMELTING COMPANY, MULLAN, IDAHO.

BY

C. E. WETHERED AND LEO J. COADY

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February, 1930

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AT THE MORNING MINE OF THE FEDERAL MINING AND SMELTING COMPANY, MULLAN, IDAHO[1]

By C. E. Wethered[2] and Leo J. Coady[3]

INTRODUCTION

This paper[4] describing the mining methods employed by the Federal Mining and Smelting Co., operating the Morning mine at Mullan, Idaho, is one of a series being prepared by the United States Bureau of Mines on mining practices, methods, and costs in the various mining districts of the United States.

The daily tonnage produced at the Morning mine, including that mined by leasers, is 1,100 tons. Mine run of ore will average 9.1 per cent lead, 5.8 per cent zinc, and 4.4 ounces silver per ton. A stull set and fill method of mining is employed.

ACKNOWLEDGMENTS

The writers are indebted to the authors of the U. S. Geological Survey Professional Paper 62, which was freely drawn upon for the early history and geology of the Morning mine.

HISTORY

The Morning and You Like veins were among the first discovered near Mullan and were originally worked independently. In 1887 the Morning mine was sold and with the purchase of the You Like mine a few years later, both mines came under the same management. In 1891 the property again changed hands; the new owners constructed a railway to No. 5 tunnel and built a 600-ton mill. The mine was sold again in 1895 and extensive improvements were made on the plant. No. 6

1 - The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6238."
2 - Mine superintendent, Morning mine, Federal Mining and Smelting Co.
3 - Chief engineer, Morning mine, Federal Mining and Smelting Co.
4 - Written under direction of H. G. Washburn, assistant manager, Morning mine, Federal Mining and Smelting Co., and consulting engineer, U. S. Bureau of Mines.

tunnel, designed to deliver the ore directly to the mill, was begun in 1900 and completed in 1906.

The mine was acquired by the Federal Mining and Smelting Co. in 1905.

GEOLOGY AND PHYSICAL CHARACTERISTICS OF THE ORE

The Morning mine is located in the eastern part of the Coeur d'Alene mining district approximately 2 miles from the town of Mullan. The vein lies on the north side of the Osburn fault, which is considered one of the major disturbances of the district.

A monzonite intrusive mass extending in a northeasterly direction from near the center of the district is considered the prime agent in the formation of the ore bodies. Numerous diabase dikes are common throughout the district.

The Morning vein lies almost entirely within quartzite known as the Revett formation, has a general strike of N. 65° W., and varies in dip from 80° NE. to vertical. The productive ore zone in the Morning mine is from 1,500 to 2,000 feet in length. The 3,450-foot level, at present the lowest level on which the vein has been exposed, is 5,000 feet below the outcrop.

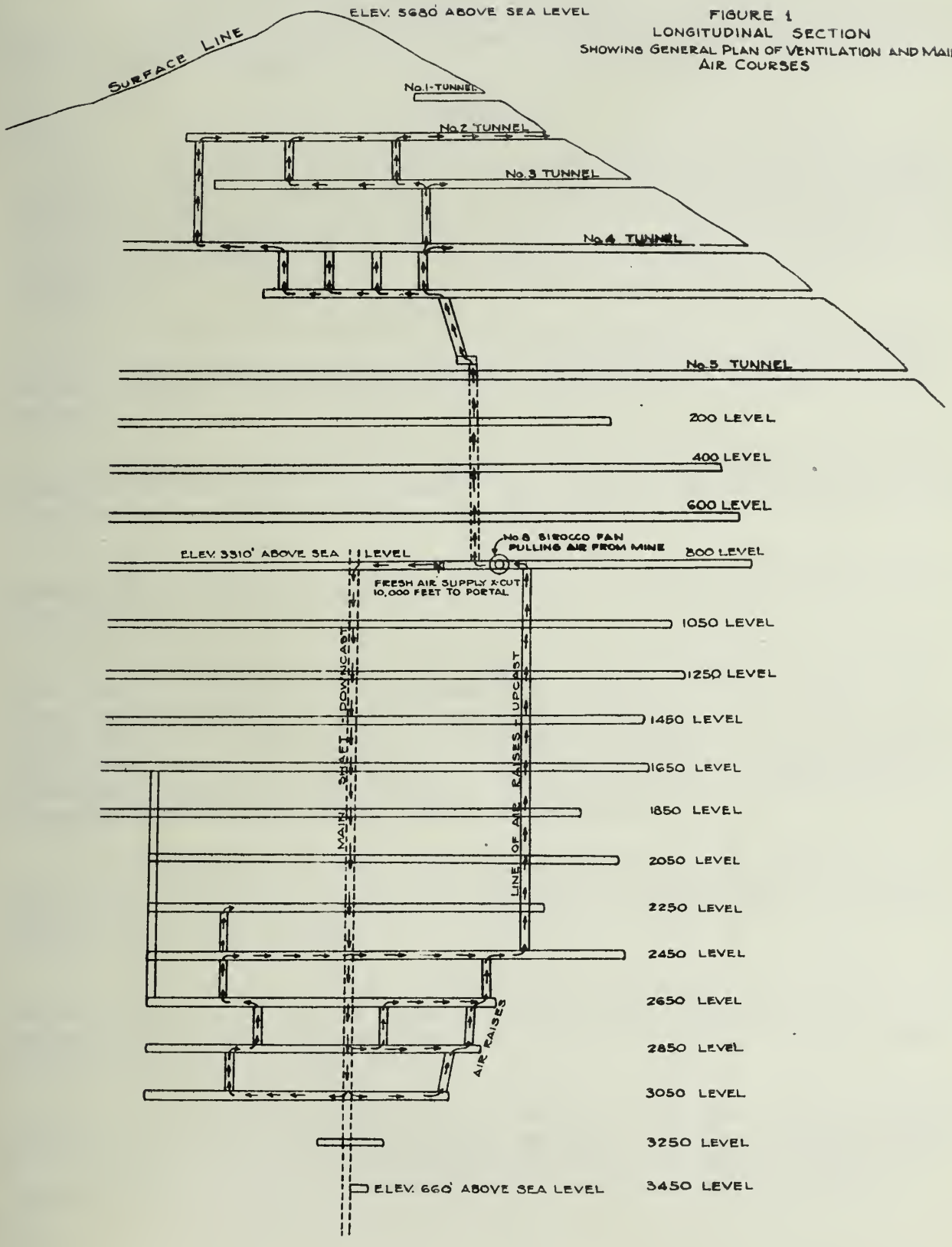
Structurally, the Morning vein is a metasomatic fissure vein in a zone of close sheeting which follows or cuts at small angles the slaty cleavage of the quartzite. The fissuring is so close and so intimately related to the general cleavage of the rocks that parts of the lodes might appropriately be called shear zones. The ore has been deposited along these zones as the filling of small fissures, thin streaks along cleavage planes, and as irregular bunches at intersections of fissures. No important fault problems are encountered, although a postmineral fault follows the vein. It is persistent and well marked at all horizons in the vein.

The stoping width of the vein varies from 6 to 30 feet and averages about 13 feet. The fissuring runs nearly parallel to the bedding of the quartzites; in some places it extends into the walls to considerable depths, whereas in others it dies out quickly after leaving the main fracture. Mineralization has taken place where the fissuring was most intense, and has entirely replaced the quartzite with ore.

In mining, the ore breaks well and large bowlders of ore are seldom encountered. Most large pieces are readily broken with a hammer. If too hard and large to be broken by hand, they are plugged with a jackhammer and blasted between shifts.

The ore is not heavy, and the timbers are not required to bear much vertical weight. The physical characteristics of the wall rock, and its tendency to swell and slab off in large platy pieces as soon as the air reaches it, cause the greatest concern in mining. This side pressure from the walls necessitates close timbering at all times. Also, if any openings in the vein, such as drifts or raises, are to be kept open, continual repair work is necessary.

FIGURE 1
LONGITUDINAL SECTION
SHOWING GENERAL PLAN OF VENTILATION AND MAIN AIR COURSES



The principal ore minerals are galena and sphalerite. The gangue minerals are siderite, barite, and quartz.

In parts of the mine, sphalerite and galena are found in a fine-grained, intimate mixture. In other parts, galena and sphalerite occur crystallized in separate bands or streaks.

EXPLORATION

Three veins have been developed at the Morning mine: the Morning vein, the You Like vein, and the Midnight vein. Other than some leasing being done in the upper workings of the You Like vein, all active mining operations at present are confined to the Morning vein. The Midnight vein produced a few thousand tons of ore near the surface but was not productive with depth and has not been worked for some time.

The ore zone in the Morning vein is practically continuous from the surface to the 3,450-foot level, which is the lowest point at which the vein has been exposed, or, as before stated, is approximately 5,000 feet below the outcrop.

The vein outcrops on the surface, and early exploration work was by open-cuts, shallow shafts, and tunnels. The location and vertical depth between tunnels is shown on the longitudinal section, Figure 1.

In an endeavor to locate other ore bodies some diamond-drilling at various points in the mine was done in the past. No data on this work are available.

The walls of the vein are prospected by long-hole drills. A 225-pound long-hole machine with independent air rotation is the type used. This machine uses a 1-1/2-inch, round, lugged shank chuck, which is reduced by a sleeve ahead of the water swivel to 1-1/4 inches in order to use the standard 1-1/4-inch, round drill steel as rods. The gage of the bits starts with 3-1/4 inches and is reduced one-sixteenth inch at each change. Air consumption is 207 cubic feet per minute at 80 pounds air pressure.

Of a total of 62 holes drilled, the greatest depth obtained was 129 feet. The average depth is 46 feet. As many of the holes were not intended to go deeper than 25 or 30 feet, this average does not mean that a higher average could not have been obtained.

Drilling speed varies with the nature of the ground and depth of hole. For a deep hole, more care is required from the start, and each bit is followed by one or more bits of the same gage. A hole 100 feet deep will not average more than 12 feet per 8-hour shift.

The labor cost, including two operators, mechanical labor on broken shanks, swivels, rods, etc., is 92 cents per foot. Supplies, including power but no charge for drill steel used in new rods, is 18 cents per foot. These costs are based upon 62 holes drilled with a total footage of 2,868 feet.

The continuity of the ore zone in the Morning vein makes exploratory work comparatively simple. In opening up a new level the shaft is sunk to the desired depth, a station cut, and a crosscut run to the vein; drifts are then run.

SAMPLING AND ESTIMATION OF ORE RESERVES

The development drifts on the vein are spaced 200 feet apart vertically. At the time these drifts are run, development samples are taken every 10 feet across the vein. Assays are made for lead, zinc, and silver, and together with widths sampled are posted on sample maps. The mining widths of partly-mined blocks of ore are shown on the stope maps and are posted monthly.

Ore reserves are figured yearly for the entire mine, and are based upon the average stope widths and the properly weighted assays in development work done during the year. In estimating tonnages, a factor of 9 cubic feet to the ton is used.

Experience has shown that between levels there is little change in the grade of the ore and that stope sampling would be an unnecessary expense. Therefore, little sampling is done in the stopes during active mining, and those samples that are taken are not considered in estimating the grade of ore in partly-mined blocks.

DEVELOPMENT AND MINING

Development. - Most of the early development of the Morning mine was accomplished by a series of tunnels driven on the vein. A narrow-gage track extended from the mill to the portal of No. 5 tunnel and delivered a considerable amount of ore mined above No. 5 tunnel to the mill.

The main haulage level, known as No. 6 tunnel or the 800-foot level, was driven during the period from 1900 to 1906. This tunnel is a crosscut 10,000 feet long from the portal to the collar of the main shaft. With the exception of a small tonnage of carbonate ore mined in the upper workings, all ore produced at the mine is transported through No. 6 tunnel to the mill bin.

The collar of the main working shaft is on this level, 110 feet in the footwall of the vein. It is a vertical, four-compartment shaft and has been sunk to the 3,550-foot level, a distance of 2,750 feet below the collar. All ore mined below the 800-foot level is hoisted through this shaft. Likewise, all supplies for points below this level are handled through it.

Shaft stations are cut every 200 feet on the east side of the shaft and a crosscut is driven north to the vein. This crosscut is used for tramming timber and supplies to the stopes. The skip pocket is cut on the west side of the shaft, and another crosscut, at a distance of 100 feet from the station crosscut, is driven to the vein for ore haulage. Placing the skip pocket and station on opposite sides of the shaft eliminates traffic congestion.

From the points where these crosscuts encounter the vein, drifts are driven east and west to the limits of the ore body. Figure 2 is a plan of the 2,650 and 2,850 foot levels, and also shows the location of the shaft, stations, supply crosscuts, and skip-pocket crosscuts.

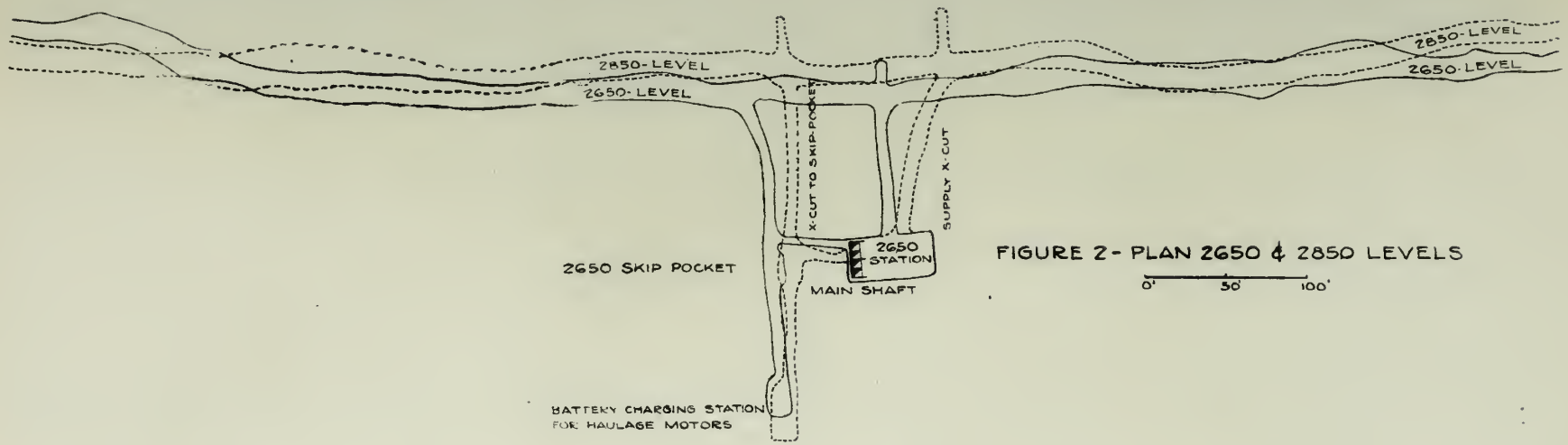


FIGURE 2- PLAN 2650 & 2850 LEVELS

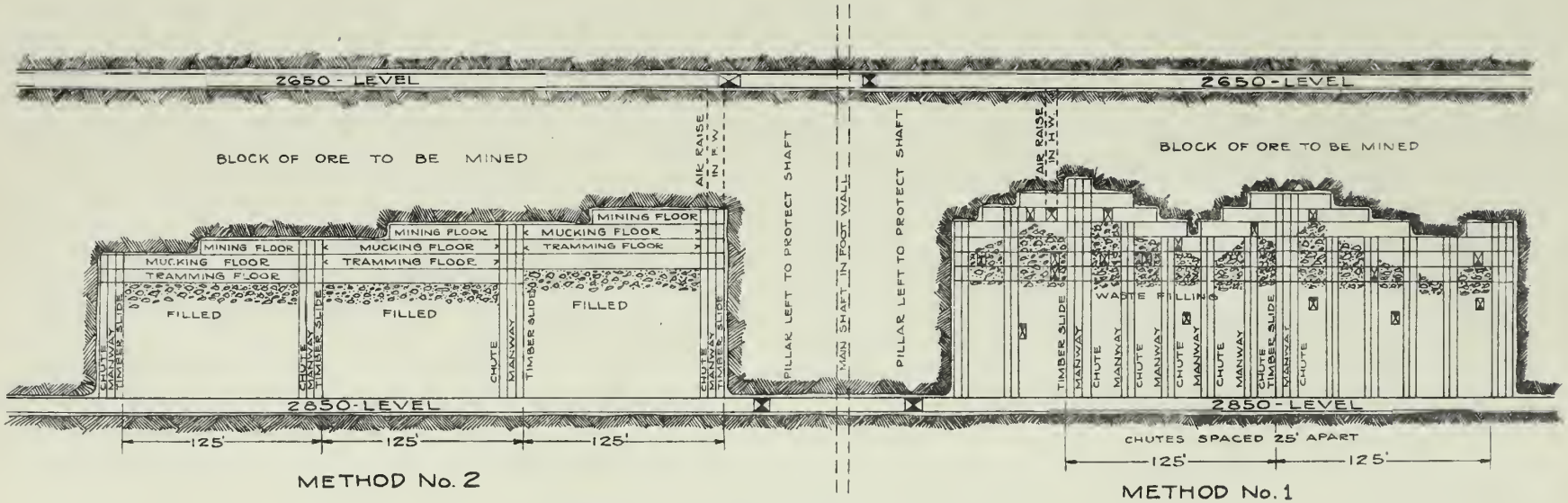


FIGURE 6.- LONGITUDINAL SECTION

0' 50' 100'

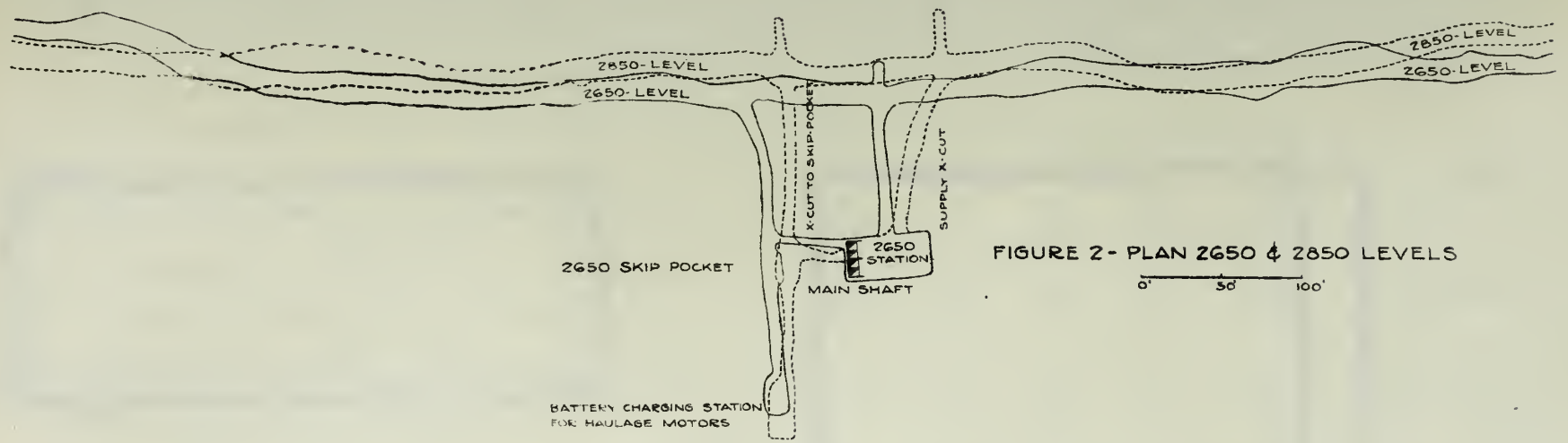


FIGURE 2- PLAN 2650 & 2850 LEVELS

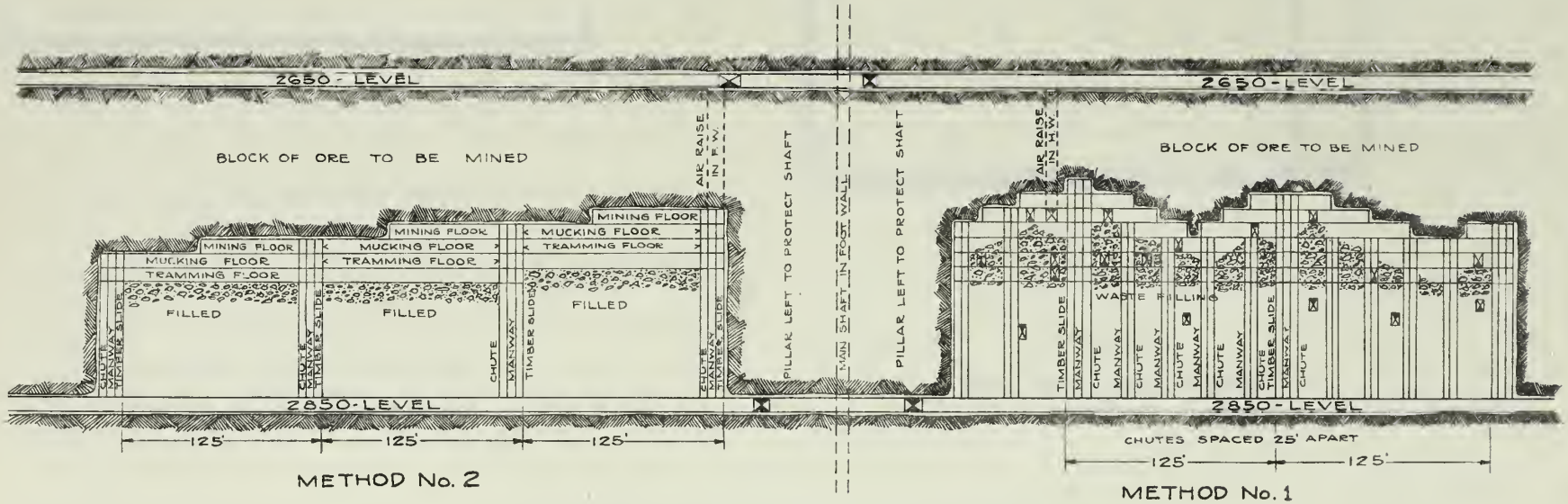


FIGURE 6.- LONGITUDINAL SECTION

0' 50' 100'



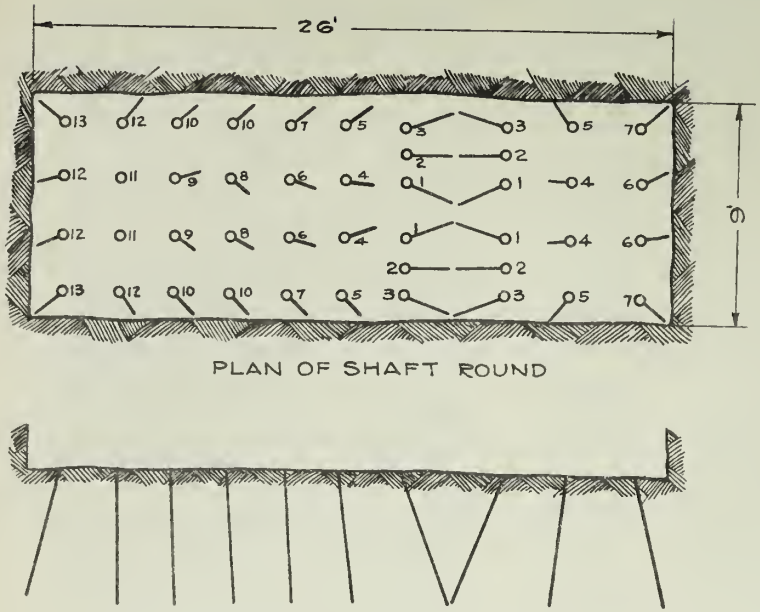


FIGURE 3

PLAN AND SECTION
SHOWING
STANDARD SHAFT ROUND

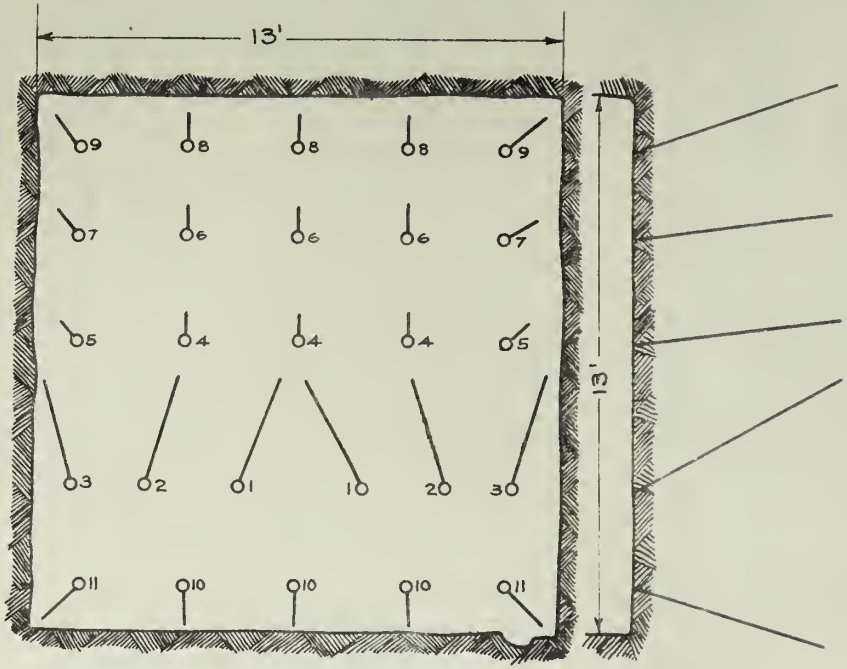


FIGURE 5

STANDARD DRIFT ROUND





As soon as the drift to the east is extended to the limit of the ore, a ventilation raise is put through to the level above.

Shaft-Sinking. - The main shaft is the only one in which any shaft-sinking has been done in recent years. It has two main hoisting compartments--a supply compartment, and a pipe compartment. The size of the shaft measured outside timbers is 23 feet 6 inches by 7 feet 3 inches. Standard 12 by 12 inch framed shaft timbers are used. The sets are 5 feet center to center, with 10 by 14 inch dividers.

Shaft-sinking operations are carried on in 8-hour consecutive shifts. Four men on each shift compose the shaft-sinking crew. The rock section broken is about 9 by 26 feet, and the cycle of operations is as follows: Assuming that a round is ready to be drilled, the shaft crew proceeds to the bottom of the shaft and immediately four drilling machines, hose, drill steel, and all tools needed are lowered to them. The machines are connected to the compressed-air line and drilling begins. A light drifting machine with jackhammer mountings using 1-1/4-inch, hollow-round, lugged shank steel is the type of machine used. Each man takes a corner of the shaft and begins drilling. As each completes his line of holes, he retreats toward the center of the shaft. To protect the holes from mud, a 2-1/2-inch hole is drilled 5 inches deep and pieces of scrap 2-inch pipe 12 inches long are driven down as collars. The hole is then drilled inside this collar. Four changes of steel with a one-eighth-inch decrease in gage of bit are used to a hole. Starters are about 2 feet long, with 1-3/4-inch gage; seconds 4 feet long, 1-5/8-inch gage; thirds, 5-1/2 feet long, 1-1/2-inch gage; and fourths 7 feet long, 1-1/4-inch gage.

Approximately 48 holes 6 feet deep, are drilled to a round. It usually requires two 8-hour shifts to drill and blast a round. Blasting is done electrically, using No. 8 delay electric blasting caps. About 3-1/2 boxes of 40 per cent gelatin dynamite is used to a round. No tamping is needed, as the powder in the holes is usually covered with water by the time the holes are blasted. The position of the holes and order of firing in the standard shaft round is shown in Figure 3.

Timbering follows the blasting of a round and usually requires a full shift to put in a set of timbers. The bottom is carried far enough below timbers so that after a round is broken the men can stand on the muck pile while timbering.

Mucking follows timbering and requires 4 to 5 shifts. The sinking bucket is handled in the pipe compartment by a small air hoist located on some level at least 200 feet above the bottom. The men shovel directly into the sinking bucket until the muck pile at the end of the shaft is considerably lower than the rest of the broken rock. To facilitate shoveling the remainder, a small air hoist is located temporarily in the supply compartment. It operates an auxiliary bucket or pan into which the men shovel and which when full is raised a few feet from the bottom and dumped into the main sinking bucket. The advantage of the pan is that it can be moved about in the bottom of the shaft and the rock can be either shoveled or scraped into it while the sinking bucket is making a round trip. With the aid of the pan the speed at which a round can be mucked out is considerably increased. A round will break about 140 tons of rock.

The sinking bucket is hoisted to the level above and dumped automatically into the waste skip pocket. From here the waste is hoisted to the main-tunnel waste bins and then trammed to the dump.

In sinking, the men are protected by bulkheads in all compartments except in the pipe compartment in which the sinking bucket travels. These bulkheads are never closer than 40 feet to the bottom, and new ones are put in when this distance approaches 100 feet.

Drifts. - Development drifts on the different levels vary somewhat in size, depending upon the width of ore encountered. The usual rock opening is 13 by 13 feet. These openings become the main haulage drifts during active mining on the levels. All drifts are timbered with the standard drift set shown in Figure 4. Caps are made from stulls about 2 feet in diameter that have been slabbed on one side. The caps vary in length, depending upon the ground broken, and are blocked to the walls with at least an 18-inch basket heading, as shown in Figure 4. A 3-inch open space is left in the heading. When side pressure from the walls closes this opening, it is time to change the blocking if the cap is to be saved.

Drift sets are spaced 5 feet 2 inches center to center. During drifting operations the men working at the face are protected by stringers covered with 3-inch lagging. These stringers extend out over the last cap to within a foot or so of the face. The other end is blocked down from the back. As the drift advances the blocking is removed and the stringers are moved ahead and reblocked.

As previously stated, when the crosscut from the shaft encounters the vein, development drifts are started east and west. Thus two working faces are available, and the broken ore in one face can be removed with a mechanical loader while drilling or timbering is being done in the other. Four miners, two on each shift, do the drilling and timbering. The workmen who break the ground and do the timbering are given day's pay plus a bonus on the advance made. One motorman and helper on each shift tram the loaded cars out and return the empty cars to the shovel, but they do not share in any bonus. The operator of the mechanical shovel (one on each shift) is paid day's pay plus a bonus on the number of cars loaded.

About 25 holes, 6 feet deep, are drilled to the round, using two regular drifting machines at the face. The explosive used is 40 per cent special gelatin dynamite. The standard drift round is shown in Figure 5. The figures on the sketch indicate the order of firing the holes. The shoveling machine is moved from one drift face to the other.

Raises. - Ventilation raises are driven in either the hanging wall or footwall of the vein (usually the footwall), and are timbered only where bad ground makes it necessary. In order to keep in the wall as far as possible, the first 100 feet of the raise is given a slope of 45° away from the vein and the last 100 feet is sloped at the necessary angle toward the vein so that it will connect with the level above. An air raise is about 5 by 10 feet in cross section and is timbered only where the ground is bad. A good electrically lighted ladderway is maintained, and travel through it is easy.

Three compartment raises in the ore with a timber slide, manway, and chute are carried at 125-foot intervals. They are carried just in advance of the stope back, and are raised only a few sets at a time. The spacing of the raises at 125-foot intervals with no intervening chutes is a method recently developed. Formerly all chutes were spaced 25 feet apart, and a manway was carried in the adjoining set. The two methods of spacing chutes are shown in Figure 6.

A raise crew consists of two men on each shift. Two regular wet stoping machines with 1-inch, quarter-octagon, hollow steel are used for drilling. About five sticks of 35 per cent powder are used to each hole. No tamping is employed. (Fig. 7.)

Stoping. - As stated before, the swelling nature of the ground and gradual breaking down of the walls along innumerable shear planes results in heavy ground and expensive timbering. The repair cost of maintaining chutes spaced at 25-foot intervals is exceptionally high, and in order to eliminate this expense, the method of carrying chutes at 125-foot intervals was adopted. The two methods of stoping are shown in detail in the longitudinal section (fig. 6).

Regular drifting machines, with hollow-round steel, are used in stoping. The miner is aided by the mucker to set up his machine and then drills and blasts the round. A stope round consists of 10 to 20 holes, depending upon the width of the ore. The holes are drilled about 5 feet deep. Five or six sticks of 25 per cent gelatin dynamite are used to a hole. No tamping is employed. Nearly all blasting is done on night shift, which gives more time for powder smoke to get out of the stopes between shifts.

The ore breaks well, and the few large bowlders encountered are readily broken up with a hammer. In the present method of mining, broken ore falls to the mucking floor one set below and is shoveled by hand into a car on the tramming floor the next set below through movable steel slides. From here it is hand-trammed in 1-ton cars to the chute and dumped. The steel slides are convenient in that they can be quickly moved from one set of timbers to the next. In this way the slide is always close to the pile of broken ore. The slide is 5-1/2 feet long and 2 feet wide. The sides are made of a triangular piece of steel 3 feet wide at the bottom, and tapering to a point at the top. The top of the slide rests upon a cap. The lower end is suspended under the next cap and is held in position by chains hooked over spikes driven in the cap. The slide is so designed that when in this position the lower end just clears the car. It is not necessary to use side boards on the car when loading from the slide.

As soon as the face on the mining floor has advanced 125 feet the rails, tram car, and steel slides are moved up one floor. The rails are relaid, a new mining floor is started, and the cycle of operations is repeated. The top of all chutes in the stope have grizzlies through which the ore is dumped. They are made from old rails cut in 4-foot lengths and spaced 10 inches apart.

The regular drift set is used in timbering the stopes (figs. 4 and 7). A stope set consists of two round posts 8 feet long and one round cap (slabbed

on one side) varying in length from 8 to 16 feet. The length depends upon the width of the ore at that point. Two girts, 10 or twelve 3-inch lagging, blocks, and wedges complete the set. The overall height of a stope set is 9 feet, and from the center of one cap to the center of the next is 5 feet. The timber is hoisted up the timber slide of the raise to the mining floor by a small air hoist. With the aid of additional men the timber is transferred and placed in position. Timbermen working in pairs make the preparations for a set of timbers and block the set in place; several pairs of timbermen may be working in the same stope.

All supplies are also hoisted up the timber slide of the raise to the stopes and the dulled steel is lowered. Waste for filling the stopes is obtained by hand-sorting in the stopes or by driving waste raises or crosscuts in the walls. The waste raises are driven on about a 45° slope. Waste crosscuts are driven horizontally and at right angles to the strike of the vein. Scrapers are used for removing waste from the waste crosscuts to the stope. Waste filling in the stopes is not leveled off but is allowed to lie on the angle of repose. No waste for filling is transferred from one level to another through the shaft. If waste from development can not be used on the level on which it is made, it is hoisted to the main tunnel and trammed to the surface waste dump.

Ventilation raises are often put through from a stope to the sill above. When such a raise is contemplated, enough unfilled space is left in the stope to take care of the waste made in the raise. The raise is then started from the top of the stope, and driven in the footwall to the level above.

The procedure above described is continued to about the eighteenth floor and the stope is filled. The remaining floors are mined in vertical sections from the above level down. This is known as "pulling sills."

TRANSPORTATION

A 4-1/2-ton storage-battery locomotive equipped with 78 wet-battery cells is used to haul ore on each of the main working levels. Ten 28-cubic-foot side-dump ore cars make up a train. The cars are loaded directly from the chutes by the motorman's helper. The motorman spots the cars. It requires but a few minutes to load a train, as the ore runs freely from the chutes. The 10-inch grizzlies on the top of all chutes prevent large bowlders from getting into the chutes and greatly aid the speed with which the chutes can be drawn. The maximum haul on any level is approximately 900 feet.

The ore on each level is dumped into a skip pocket having a capacity of 200 tons. The ore is loaded from the skip pockets into 4-ton skips and is hoisted to the top station or 800-foot level. On the top station the skips dump automatically into a 200-ton ore bin. The bin also has a compartment for waste.

For loading the skips, each skip pocket has a vertical air-lift gate which is operated by the skip tender.

The skips in the two main hoisting compartments are operated by a double-reel hoist driven by a 600-hp. electric motor. The hoist normally

operates in balance with a rope speed of 1,300 feet per minute. The capacity of each skip is 4 tons. A 1/2-inch by 6-inch flat cable is used.

The cage in the third compartment, which is used for men, timber, and supplies, is operated by a 300-hp. electric hoist.

From the top station bins to the 1,000-ton mill bins is a distance of 10,000 feet. A main-haulage train in this tunnel is composed of sixteen 4.2-ton, bottom-dump cars. As each car weighs 2 tons, the combined weight of train and ore is, roughly, 102 tons. These main-haulage trains are handled by 6-1/2-ton, electric trolley locomotives operating on 550 volts. The round trip requires about 45 minutes.

As the ore train passes through the head house it is weighed, in sections of four cars each, on a scale equipped with a recording beam so that a weight ticket is printed for each weighing. The empty train on its return is again weighed in corresponding sections. In this way an accurate net weight of the ore sent to the mill is obtained.

With a few minor exceptions, the only hand-tramming in the mine is done in the stopes between chutes on the regular tramming floors.

PERCENTAGE OF EXTRACTION

All ore developed is ultimately mined. It is estimated that dilution of ore by waste is 20 per cent. No waste is sorted on the surface.

WAGE, CONTRACT, AND BONUS SYSTEM

All labor is based upon an 8-hour day. Wages are uniform throughout the district and are based upon a sliding scale according to the price of lead. During 1928 when lead sold for 6 to 6-1/2 cents a pound, wages were as follows:

	Per day
Miners	$5.00
Timbermen	5.50
Timbermen helpers	4.75
Muckers	4.50
Motormen	5.25
Motormen helpers	4.75
Shaftmen	6.00

No contract or bonus system is employed in actual stoping operations. All work is paid for at day's pay according to the classification under which the work comes.

Shaft-sinking, skip chutes, station-cutting, etc., are usually contracted for at so much a linear foot or cubic foot. The main development drifts on the levels are generally driven on day's pay plus a bonus for speed. Raises and crosscuts are contracted for at so much a linear foot.

The mine superintendent has charge of all operations underground. His plans are carried out in detail by three foremen and a corps of shift bosses, none of whom shares in the contracts or participates in any bonuses.

VENTILATION

On account of the many tunnels and connecting raises of the Morning mine, natural ventilation is good on all upper levels. As depth is reached below the 800-foot level, ventilation becomes more difficult and mechanical ventilation is necessary.

The main air courses with direction of air movement are shown in Figure 1. A No. 8 fan, with a capacity of 35,000 cubic feet of air per minute, located on the 800-foot level, pulls air from the mine up through the main air raises. From the fan the air travels up through an enclosed compartment in the shaft to No. 5 tunnel and on up an air raise to the upper workings, finally finding its way to the surface.

The main hoisting shaft is downcast and the main line of east air raises is upcast. Properly-regulated doors on all the levels help the fan to maintain this direction of air movement. If fire occurs in certain parts of the mine, or if for any other reason it is desired, the direction of air movement can be reversed by closing one door and opening another at the fan. In this way, fresh air could be forced down the line of air raises and the shaft would become upcast.

All the fresh air going down the main shaft flows through the main-haulage tunnel for a distance of 10,000 feet. Small fans are located at various points on the main air courses and with the aid of flexible canvas tubing they furnish air for dead ends and out-of-the-way places. Some compressed air is also used for ventilation.

Temperaturein the mine is not abnormal. In a recent test the highest temperature recorded was 82°F., but the humidity was very high. This high humidity at the temperature of the lowest levels has an effect upon the amount of work a man can do without tiring.

FIRE HAZARDS AND SAFETY METHODS

No apparent fire hazards exist in the mine. While the mine is not considered wet, all parts are moist because of the humidity of the air. At no points in the mine are old rubbish, boxes, etc., allowed to accumulate. Standard methods of electric wiring are used and the wiring is frequently inspected.

About 35 men employed in and about the mine have had training in the use and operation of helmets. These men must be ready to answer emergency calls and are paid extra.

The mining companies of the district jointly maintain a rescue car at Wallace, Idaho. The car carries the latest rescue and first-aid equipment and is subject to call at all times. Eight helmets are kept at places in and about the mine and are periodically inspected Training in first-aid work is given at the mine to all employees who are to avail themselves of the opportunity

To aid in the prevention of accidents underground, a bonus is paid to the shift boss and foreman having the least number of accidents. The shift bosses' bonus is based upon and paid on a monthly, quarterly, and yearly record. The foremen share in the monthly and yearly bonuses only.

A safety engineer is employed at the mine.

Table 1. - Mining Costs

Name of mine: Morning mine Period covered: 1928

Tons of ore hoisted: 276,890

Underground costs per ton of ore mined

	Labor	Super-vision	Compressed air drills and steel	Power cost	Explo-sives	Timber	Other supplies	Total
Development	0.312	-	0.031	0.019	0.044	0.036	-	0.442
Mining	1.550	0.142	0.091	0.098	0.114	0.322	0.054	2.371
Transportation (underground) ...	0.369	-	-	0.060	-	-	0.101	0.530
General underground expense	0.556	-	-	0.034	-	0.059	0.049	0.698
Surface expense directly applicable to underground operation	0.159	-	-	-	-	-	0.045	0.204
Total	2.946	0.142	0.122	0.211	0.158	0.417	0.249	4.245

Table 2. - Summary of Costs in Units of Labor, Power, and Supplies

Name of mine: Morning mine Period covered: 1928

Tons of ore hoisted: 276,890

Mining method: Breast stoping with stull sets and fill.

	Development	Stoping	Total
A. Labor (man-hours per ton):			
Drilling and blasting	-	-	0.514
Timbering	-	-	1.740
Mucking	-	-	0.583
Haulage and hoisting	-	-	0.419
Supervision	-	-	0.181
Underground maintenance	-	-	0.725
Total labor underground	-	-	4.169
Tons per man-shift	-	-	1.918
Labor per cent total cost	-	-	72.5
B. Power and Supplies:			
Explosives (lbs. per ton) (25% ...	.002	.349	.351
(35% ...	.082	.273	.355
(40% ...	.187	.071	.258
Timber (bd. ft. per ton)	-	-	12.753
Power (kw.h. per ton)			
(1) Air compression	-	-	28.820
(2) Hoisting	-	-	4.351
(3) Pumping	-	-	2.262
(4) Ventilation	-	-	2.736
(5) Lighting	-	-	1.200
Total	-	-	39.369
Other supplies in percentage of total supplies and power	-	-	48.4
Supplies and power percentage of total cost	-	-	27.5
C. Percentage Total Cost	10.3	89.7	100.0

Table 3. - Detail of Development Costs in Units of Labor, Power, and Supplies

Name of mine: Morning mine — Period covered: 1928

	Development		Cross-cutting	Raising	Total all Development
	Sinking	Drifting			
Size of excavation	9x26 ft.	13x13 ft.	6x8 ft.	5x10 ft. 17x10 ft.	
Timbered or not	Timbered	Timbered	No	Timbered	
Physical properties of rock	Hard rock swelling ground	Hard swelling ground	Hard swelling ground	Hard swelling ground	
A. Labor (man hours per ft.):					
Breaking (drilling and blasting) Timbering Mucking Haulage and hoisting Supervision	72.7	28.8	8.9	11.8	24.3
Total labor	72.7	28.8	8.8	11.8	24.3
Feet per 8-hour man-shift	0.11	0.27	0.9	0.67	0.53
B. Power and Supplies (per ft.):					
Explosives (lbs. per ft.)	44.0	20.0	11.3	14.0	17.3
Timber (bd. ft. per ft. advanced)	419.0	149.1	6.9	68.3	100.8
Other supplies (per cent total power and supplies)	27.7	20.0	24.6	18.4	22.6
C. Labor (per cent total cost)	71.7	70.7	68.9	66.8	70.3
Power and Supplies (per cent total cost)	28.3	29.3	31.1	33.2	29.7

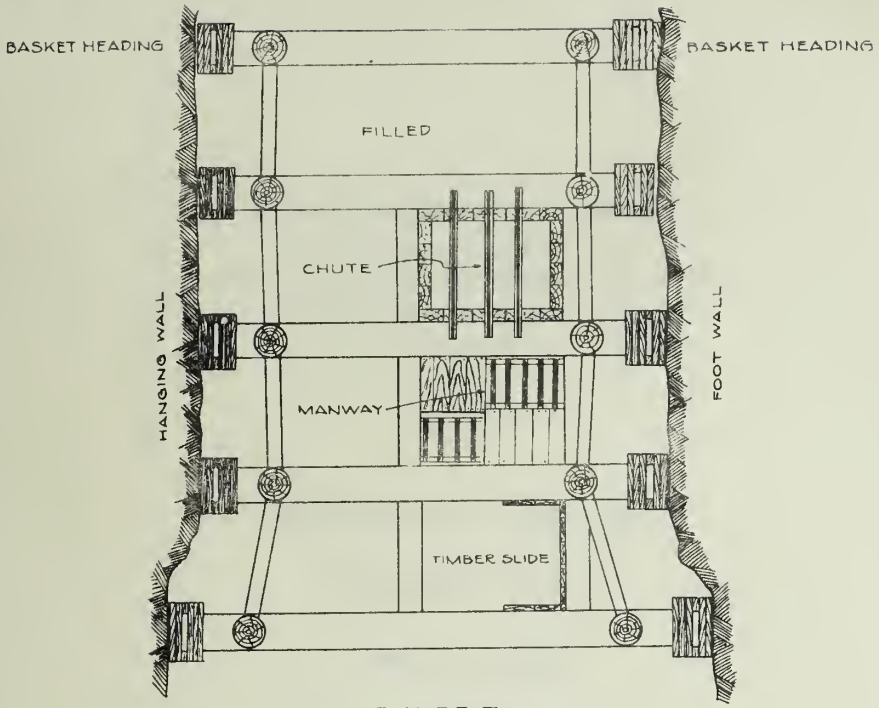


FIGURE 7
PLAN OF STOPE TIMBERING

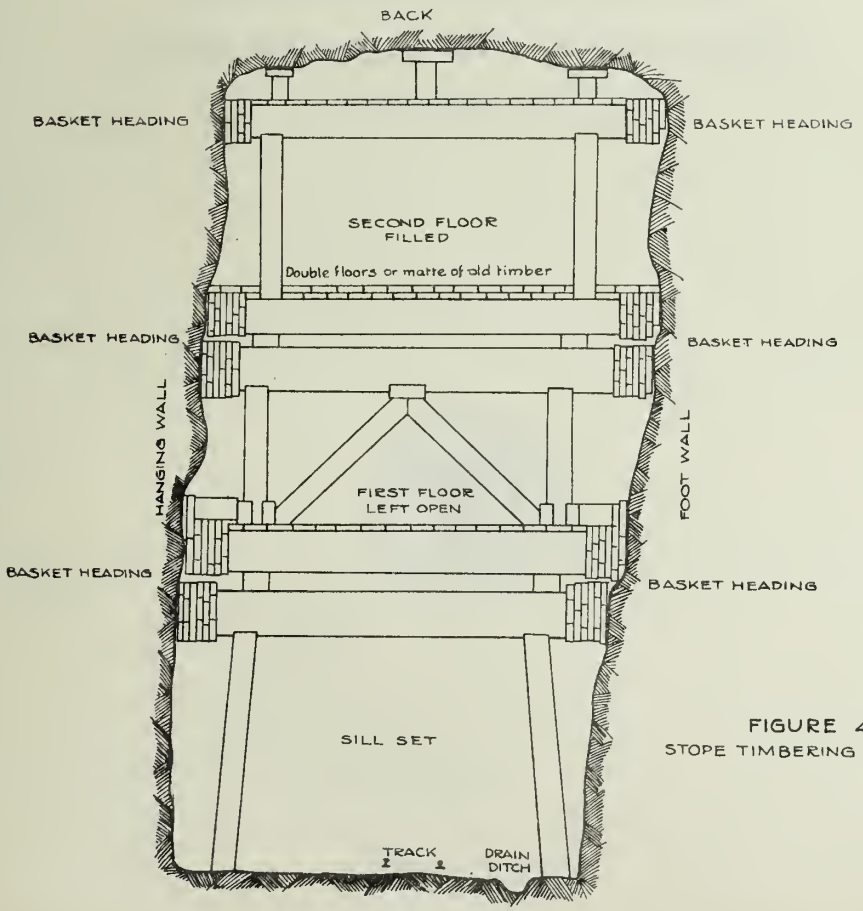
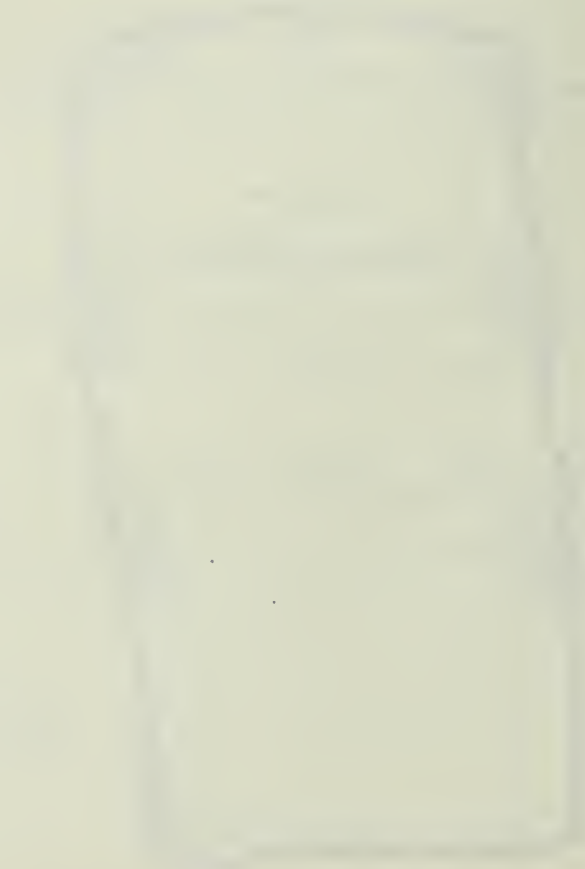


FIGURE 4
STOPE TIMBERING IN ELEVATION





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MINING METHODS AND COSTS, AMERICAN ZINC CO. OF TENNESSEE, MASCOT, TENN.

BY

HARLEY A. COY

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February, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS, AMERICAN ZINC CO. OF TENNESSEE, MASCOT, TENN.[1]

By Harley A. Coy[2]

INTRODUCTION

This paper, describing the mining methods and practice at No. 2 mine of the American Zinc Co. of Tennessee at Mascot, Tenn., is one of a series of papers issued by the Bureau of Mines presenting the mining methods and costs in various districts throughout the United States. Its purpose is to summarize methods recently put into practice whereby increased efficiency has been effected. These improvements have to do principally with mill-hole mining, slushing, contract labor, and bonuses. In general, this paper supplements earlier articles and brings them up to date.

ACKNOWLEDGMENTS

The author acknowledges the assistance of M. H. Newman, geologist of the American Zinc, Lead & Smelting Co.; of William Black, assistant superintendent of mines; of P. M. Arthur, director of personnel; and of the engineering staff. The paper has been approved by H. I. Young, vice president and manager of operations.

LOCATION

The mines of the American Zinc Co. of Tennessee are situated at Mascot and at Jefferson City, in the Holston River Valley of east Tennessee. Mascot lies 14 miles northeast of Knoxville, while Jefferson City is 18 miles northeast of Mascot. Both are stations on the Southern Railroad.

HISTORY

Before 1900 an unimportant output of oxidized ore was obtained from shallow pits at the outcrop; in 1900 a shallow shaft was sunk to sulphide ore; and in 1903 a second shaft, also for sulphides, was put down 200 feet.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6239."

2 - One of the consulting engineers, U. S. Bureau of Mines, and superintendent of mines, American Zinc Co. of Tennessee.

The American Zinc Co. of Tennessee started prospect drilling at Mascot in 1911. This drilling located and outlined the ore body that later was developed by the No. 1 shaft. No. 1 shaft is 365 feet deep, with its haulage level at 280 feet. Later No. 2 ore body was discovered by drilling about one-half mile east of No. 1 shaft. No. 2 shaft is 612 feet deep, with its main haulage level at 520 feet. The output of No. 2 mine originates in this ore body and in its extensions.

The Mascot section of Holston valley is agricultural. Mining properties are assemblages of farm lands. Their outlines are described by mete and bounds. Title commonly is held in fee. Leasing and purchase of mineral rights, while in effect, are not common.

GEOLOGY

The ore is present both as sulphide and carbonate. The sulphide type only is mined by the American Zinc Co. Little carbonate ore appears to be present at these particular properties. Where found it is in the form of small pockety deposits in clay at the sulphide outcrop.

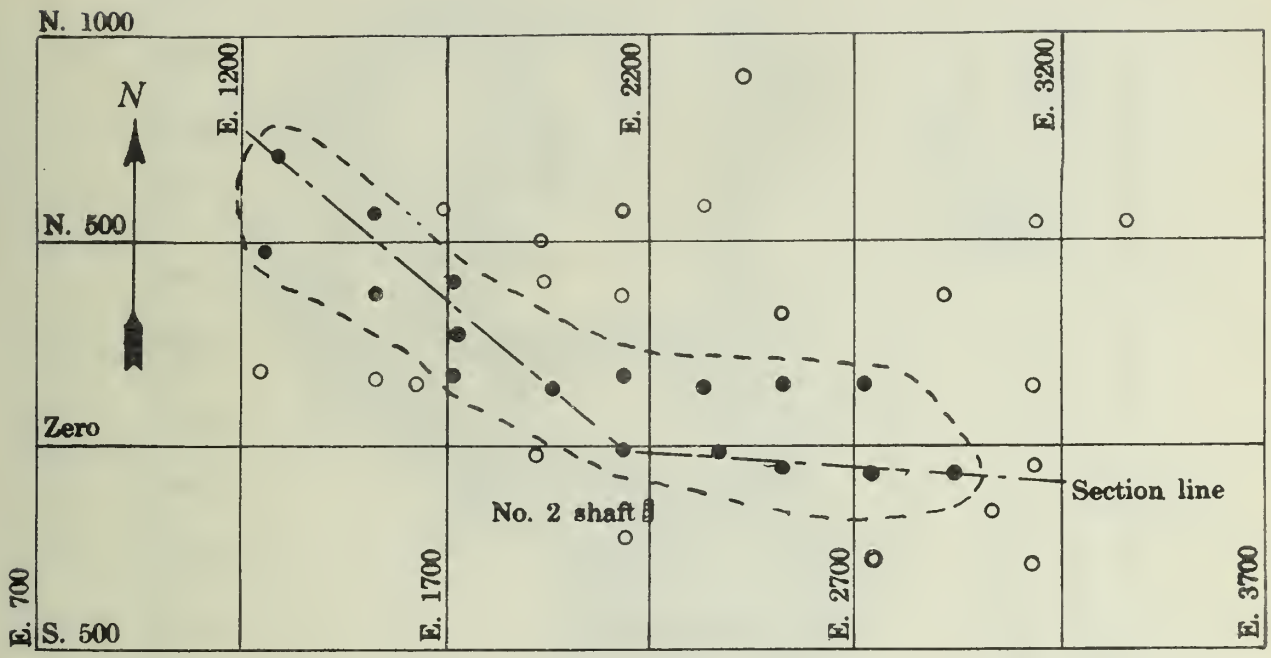
Mascot is within the area covered by United States Geological Survey Folio 75. The rock formation as classified therein is Knox dolomite. At Mascot this formation strikes 50° to 75° east of north and dips 18° to 22° southeast. The ore bodies fall at an horizon about 1,000 feet structurally below the top of the Knox and as far as known are confined to the beds of that horizon. No faults of consequence have been found in the mine, although faulting is known to occur in the locality.

The rock of the ore horizon is dolomitic limestone and limestone. In large part the ore occurs as veinlets and seams of sphalerite with secondary dolomite in the dolomitic limestones. Ground so mineralized ordinarily will not stand long unsupported. On the other hand, unmineralized dolomite will stand almost indefinitely, even with spaces of as much as 100 feet or possibly more between supports. Locally, occasional thin shale partings between beds so weaken the ground as to cause it to fall, but such partings are not common. When they occur near the top of the ore the insecure ground usually is taken down, and some overlying bed is carried as a roof. Particular care is used in establishing the roof in secure material. After the roof once has been established it is left undisturbed.

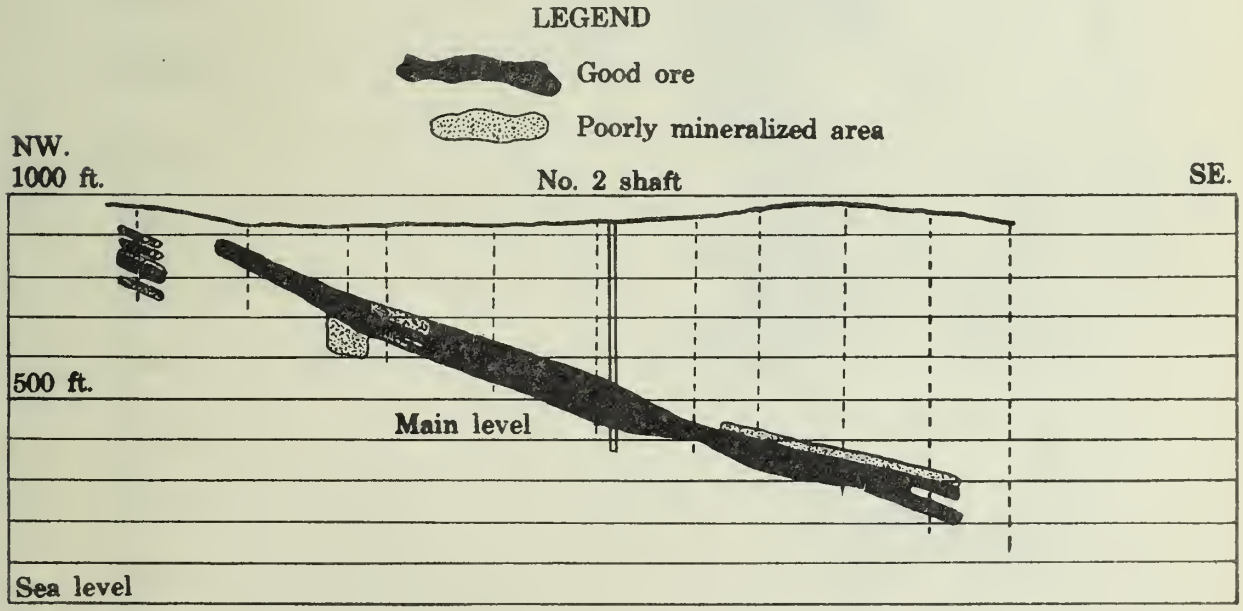
The minimum angle of gravity flow for broken ore is slightly more than 45°. By proper placing of the holes and by care in shooting, however, ore may be made to deliver at somewhat flatter angles.

PROSPECTING AND EXPLORATION

Exploration in advance of mining is by surface drilling, both churn and diamond drills being used. Holes are not located geometrically. Additional holes to insure proper mine development are drilled from time to time as needed. Churn-drill cuttings and their insoluble residues are mounted on cards for study. The final logs of churn-drill holes are built up from these cards. All of the bailings from each 3-foot advance after ore has been reached are sampled and assayed. Specimen cores from diamond-drill holes are filed for



PLAN



SECTION

Figure 1. — No. 2 ore body as outlined by surface drilling at time of sinking No. 2 shaft

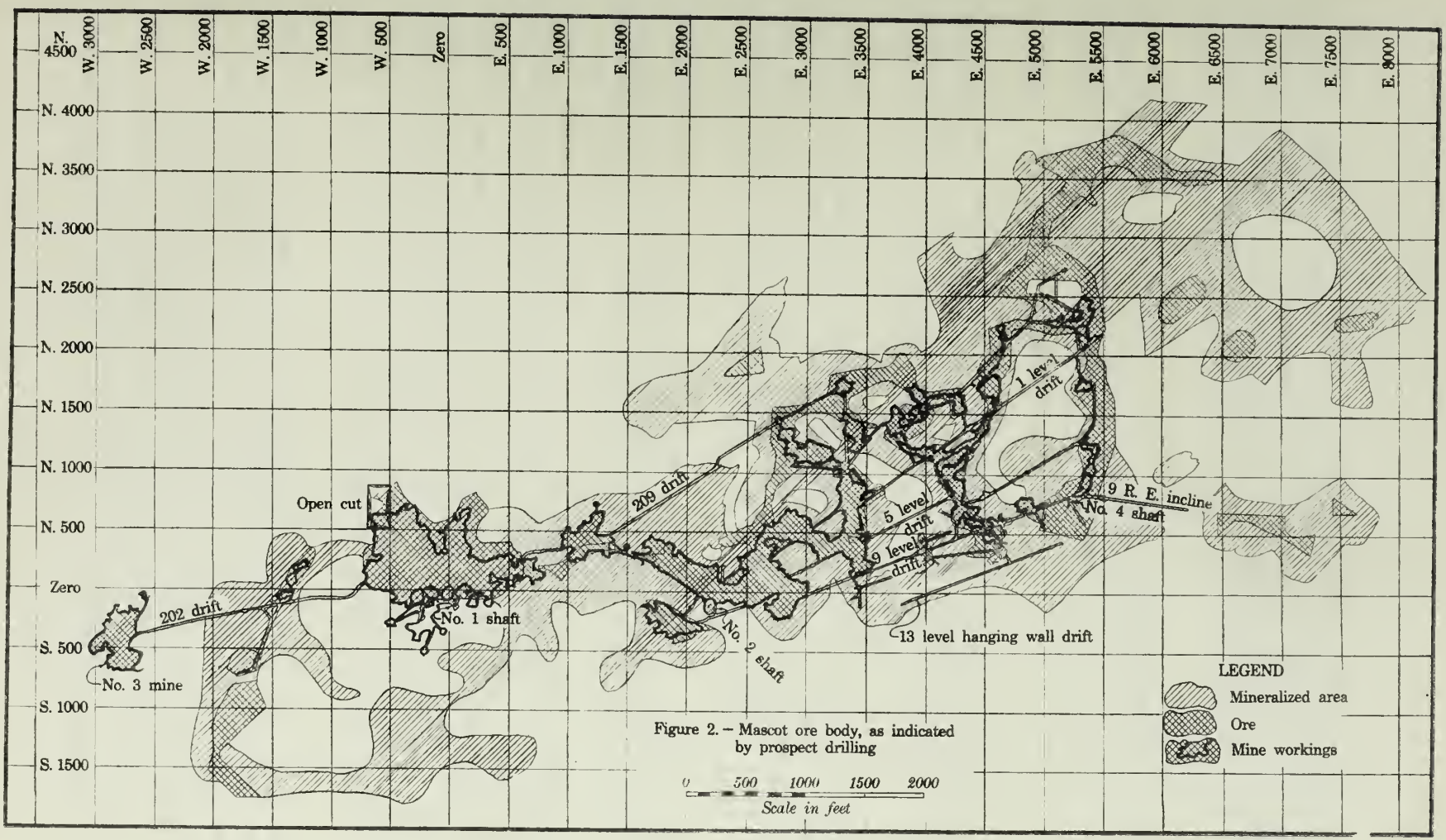


Figure 2. – Mascot ore body, as indicated by prospect drilling

formational records. In ore ground all core is included in samples taken for assay. Drifts, crosscuts, and raises seldom are driven to obtain information on ore alone. When such developments fall in mineralized ground grab samples are taken after each round has been shot. An underground diamond drill is in use continuously. Frequently drifts or crosscuts are driven to provide drilling stations for this machine. The drift-diamond drill type of prospecting is used principally in the deeper parts of the mine.

Underground sampling is confined to grab samples from developments, as above; to core from the diamond drill; and to cuttings from test holes by machine drills in roof of stopes or mill holes. Grades at working faces are estimated.

DEVELOPMENT

No. 2 mine was developed from a single vertical shaft 612 feet in depth. The main haulage level was broken off at 520 feet; the loading station for skips is at 582 feet. A vertical 6 by 6 foot raise 81 feet in height connects the bottom of the shaft with the No. 5 level and provides a draw-off for the spill incident to skip-loading.

No. 2 shaft is a square set shaft, and has four compartments. The top 48 feet - from surface to rock - is concreted. Three of the compartments, one cageway and two skipways, are 5 feet by 5 feet 6 inches. The fourth compartment, which carries the discharge columns, power lines, signal lines, and ladderway, is 3 feet by 5 feet 6 inches. Shaft timbers are of 8 by 10 inch white oak. The rock requires no timbering for support.

The shaft is in the hanging wall and south of the main No. 2 ore body. When the shaft site was chosen, prospect drilling had determined the outlines of this particular ore body, as it appears in Figures 1 and 2. Other near-by ore bodies were developed later.

A short crosscut connects the shaft station with the ore body. From the end of this crosscut drifts were driven east and west in ore near and parallel to the hanging-wall side of the ore body. From the ends of each of these drifts inclines were driven. The west incline is 600 feet long and rises from the end of the west drift on a slope of 24°, and the east incline starts at the east end of the east drift and follows a slope of 22° downward. Each incline followed closely the upper and south edge of the ore body. From these inclines drifts were turned off in ore at vertical intervals of 50 to 75 feet. Open stopes were developed from these various drifts and were mined by heading and bench methods. The ore was hand-loaded.

In the earlier development all of the haulageways were established on the hanging-wall side of the ore horizon. The general haulage scheme now in use includes these hanging-wall drifts and the two inclines. As new ore bodies were developed, it was found that mining practice could be improved by using underground glory holes or mill holes, as they are termed locally. Mill-hole development involves driving a raise from a point below the ore, and for this reason later haulageways have been established in the footwall.

In the early operation of the mine open-stope mining was supplemented by sub-

leveling, all of the ore being hand-loaded. More recently mill-holing and slushing have in large part replaced these earlier methods. A comparatively small percentage of the output is hand-loaded. Figure 3 illustrates the various steps in the development of these processes. Reading from right to left on this illustration the various stages are: A, Prospecting at low elevations is by crosscut and diamond drill. B, In early mining days, after the toe of an open stope reached the footwall sublevel mining cleaned up such ore as remained between main haulageways. C, Shovel stopes are opened up from crosscuts passing through the ore body. D, After the mill hole reaches the angle of repose for broken rock, such ore as remains is slushed to the mill-hole raise. E, Typical example of mill hole before slushing equipment is installed. F, In thin ores at considerable distances above the haulageway mill-holing is supplemented by slushing. G, Mill-holing a thick ore body a considerable distance above the haulageway.

A mill hole is developed by driving a raise either from a haulageway or from a crosscut in the footwall. It is driven through the footwall and ore to such a height as to intersect the horizon that has been predetermined as the roof of the individual working place. A chute is installed at the draw point, and the raise is reamed from 5 by 5 feet to 10 by 10 feet. A drift or incline, as the case may be, is driven from some near-by working place to intersect the top of the raise, its purpose being to provide access for men and pipe lines to the top of the raise.

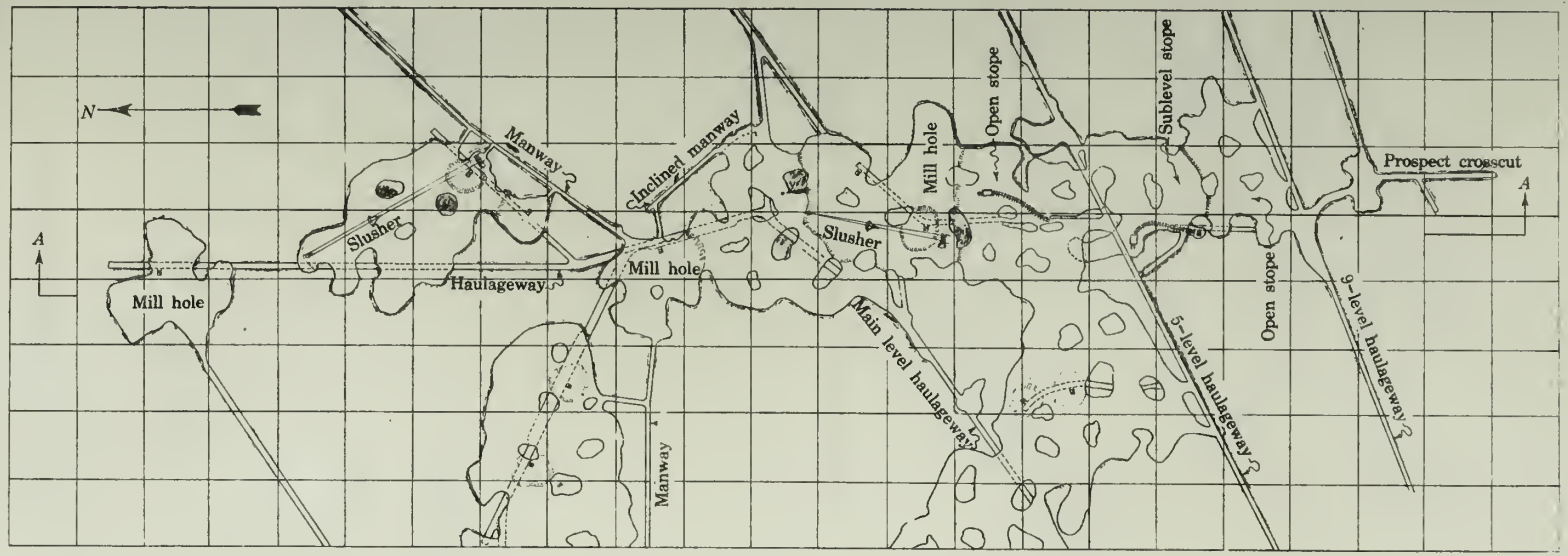
After mining of the mill hole has advanced its heading to a point so far from the raise that the ore will no longer deliver to the raise by gravity, either a second raise is driven to intersect a side of the mill hole or a drag is installed to slush the ore to the original raise. The choice of method depends upon the height of the bottom of the ore above the haulageway and upon the thickness of the ore. Where it becomes necessary to mine from open stopes with hand-shoveling it is customary to attack the ore from the hanging-wall side and to mine the ore by the heading and bench system. Figure 4 shows the location of holes, number of cartridges used, and the order of firing. Heading holes vary from 6 to 10 feet, and bench holes are seldom drilled over 10 feet. Thirty per cent low-freezing gelatin dynamite is used in shooting all holes.

All main haulageway drifts are driven to 8 by 8 foot cross section. All crosscuts are driven to 7 by 7 foot cross section. (See fig. 5, which shows the depth of holes, number of cartridges used, and order of firing.)

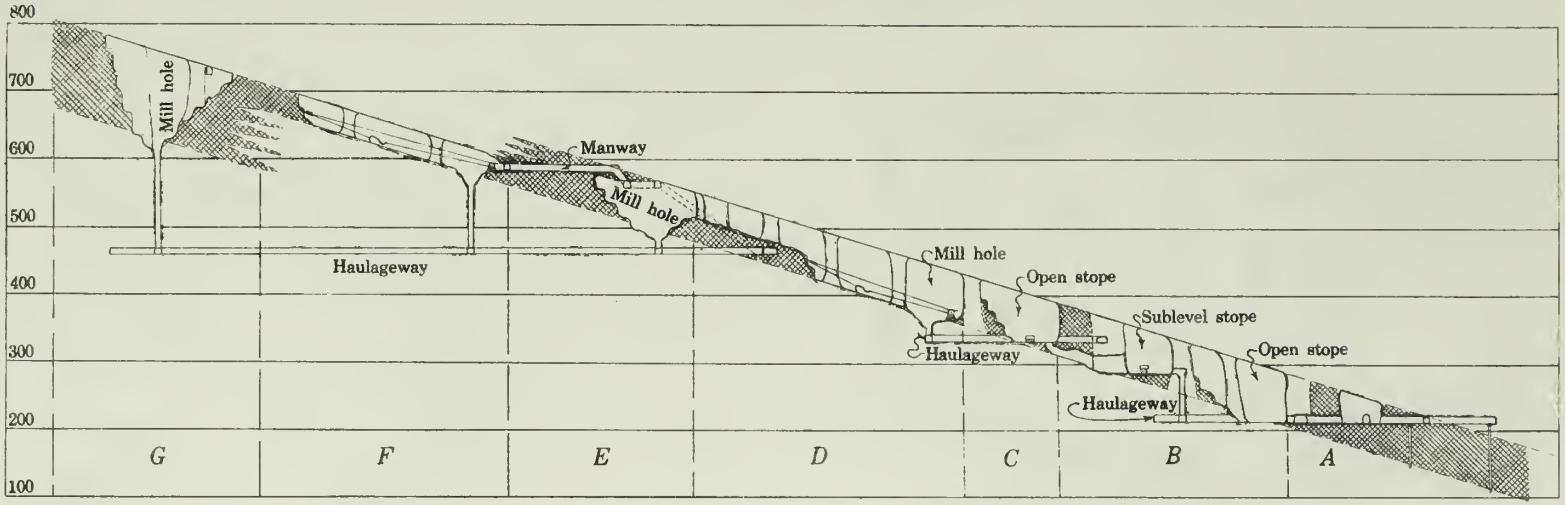
In so far as possible track grades in all drifts and crosscuts are held to 0.50 per cent, the grade being in favor of the loads. On main haulageways the minimum curve has a 75-foot radius. Few curves with less than a 50-foot radius are used in stopes. In some few hand-loading stopes curves with a radius of 25 feet are used. Standard trackage is of 40-pound rails and 24-inch gauge.

STOPING

The total production is mined by the heading and bench system. Eight-foot headings are cut immediately under the bed that has been chosen for the roof. In mill-holing



PLAN OF A PORTION OF NO. 2 MINE



SECTION *A–A*

Figure 3.– Development of mining processes

0 100 200 300 400 500

Scale in feet

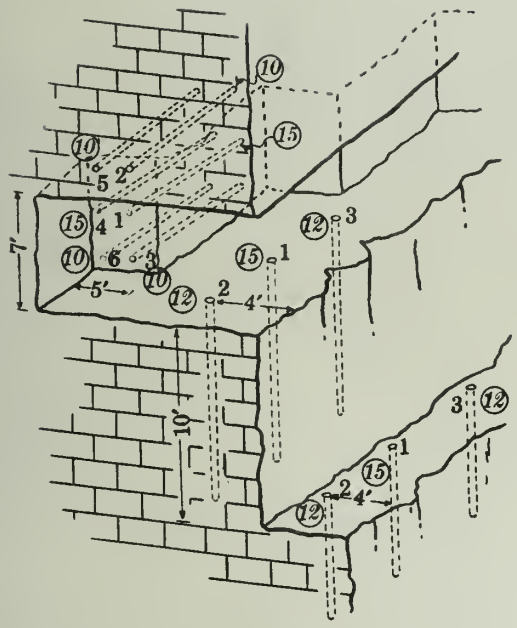


Figure 4. — Heading and bench method

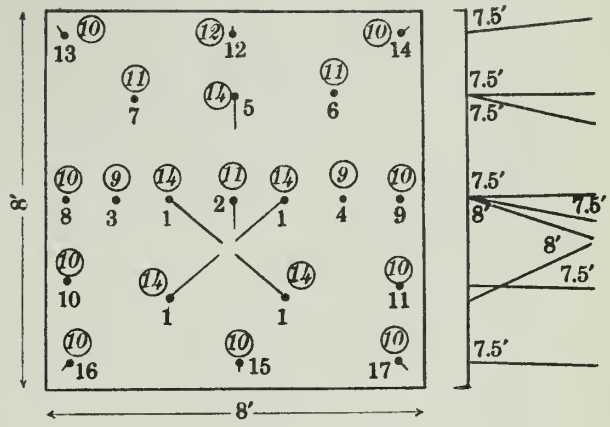


Figure 5. — Standard drift round

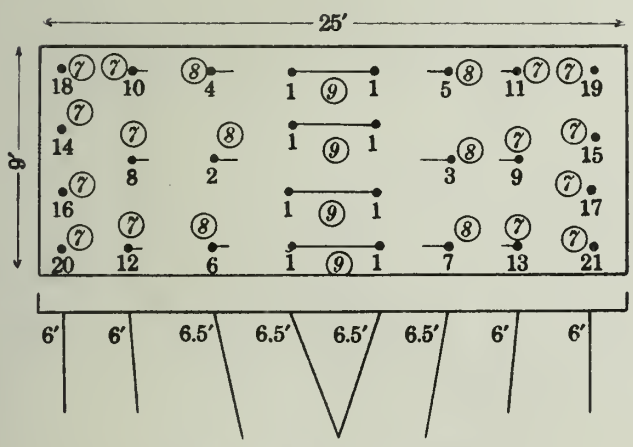


Figure 6. — Standard shaft round

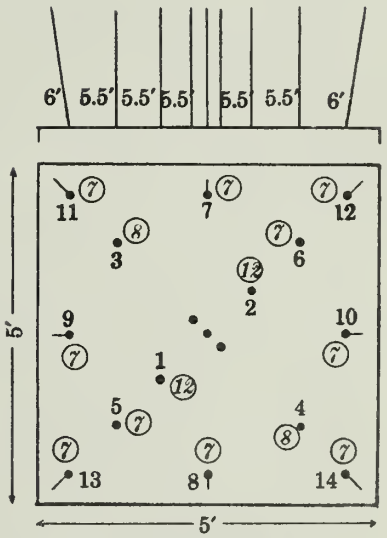


Figure 7. — Standard raise round

and in open-stoping the system used is the same, except that in developing a mill hole the mining spreads out in all directions from the top of the raise, while in open stoping the attack is entirely from one side. In both methods care is taken in spotting pillars so that such a maximum spread between pillars may be taken as is consistent with maintenance of good roof conditions.

The ore bodies vary in thickness. They occupy a low dipping position, with the result that the bottom of the ore stands at varying heights above haulageways. These two factors determine whether the ore shall be developed for mining by mill-holing or by slushing. Where the thickness of ore is sufficient and the developments required are not excessive mill-holing is preferred, but where long drifts and high raises are required, and where the thickness of the ore does not justify so much development slushing is being used more and more. The first developments looking toward slushing are identical to those of mill-holing, in that a raise is driven as in mill-hole development. Regardless of the height of ore the first mining is by mill-holing; slushing follows. In the older mill holes, where height above haulage is unfavorable to the driving of secondary raises for further mill-holing, the outlines of the original mill hole are extended by slushing.

No reserve of broken ore is stored in stopes or in mill holes. Any ore that may be left in mill holes at the end of the day shift is drawn off on the following night shift. It is important for the maintenance of grade that the working faces be uncovered each day, in order that foremen may judge it before ore for the current shift is broken.

DRILLING AND BLASTING

All drilling and blasting (except development) is on day shift. The drilling and blasting of all development work, which includes drifting, raising, reaming, slabbing, etc., are done on night shift. All development rock is shoveled on the day shift. The night-shift crew, except men working on development, consists of chute pullers and shovelers. Approximately one-third of the mine's production is moved on this shift and the remainder on the day shift. All rock from development work is moved with the ore on the day shift and goes to the mill with the ore, no sorting of waste being attempted.

A day's work underground consists of two 9-hour shifts. The night shift relieves the day shift at 4 o'clock in the afternoon. During the period between 1.45 and 6 a.m. no work, except development drilling and shooting, is being done. About 12 hours is required for the drilling out and shooting of a standard development heading. This cycle of operations allows members of the night shift to get their rest during the early part of the morning, a desirable arrangement during the summer months.

Denver dreadnought hammer drills, model 60, mounted and unmounted, are standard equipment in stope and mill-hole work. Ingersoll model 248 and Denver model 7 drifters are used in development drifting. Ingersoll B.C.R. 430 jackhammers are used in blocking in stopes, and in mill holes and at chutes. Ingersoll model B.C. 21 stoping machines are used in raise work.

One and one-quarter-inch hollow round drill steel with Carr bits is used entirely in drifting and stoping. Seven-eighths-inch solid hexagon steel with cross bit is standard

equipment in raising. All blocking machines use seven-eighths-inch hollow hexagon steel with carr bit.

The placing of holes for drilling drift, shaft, and raise rounds appears in Figures 5, 6, and 7. Figure 5 has already been discussed. Figure 6 shows a standard shaft round, with the depth of holes, number of cartridges used, and order of firing. Figure 7 pictures a standard raise round, showing the depth of holes, number of cartridges used, and order of firing. Mill-hole raises are driven 5 by 5 feet and reamed to 10 by 10 feet. Thirty per cent low-freezing gelatin dynamite in 1-1/8 by 8 inch sticks (extra plastic) is standard explosive for all work. Forty per cent dynamite of the same grade is issued to development contractors for use in shooting drift cuts in unusually hard-breaking ground.

LOADING

All shoveled ore from stopes is loaded on a ton-contract basis. Loading in development drifts is on a contract price per foot of advance, the contract varying with the cross section of the drift. Stope loaders are required to load 13 cars before leaving the mine. Three contract-shoveling rates are in use, their application depending upon shoveling conditions in individual stopes.

Slushing hoists in use are electrically driven, power being obtained from the 240-volt trolley circuit. Units of 25 hp. are used in stopes under average slushing conditions. Hoists of 10 to 15 hp. are desirable in clean-up work. A hoist of the 2-speed (mechanically controlled) change type on the load line is being tried out. It has been found to be particularly desirable in rough material. Slushing hoes in general use are constructed from manganese plates.

UNDERGROUND TRANSPORTATION

All main-line haulage is by trolley. Locomotives are of standard 6-ton design. Some are equipped with gathering reels. Trolley wire is of the figure 8 type. It is placed outside of the rail line and is supported by iron pipe hangers from the roof (fig. 8). These supports are spaced 10 feet apart on tangents, closer on curves, as required. All trolley wire is guarded on the track side. At turnouts and crossovers it is guarded on both sides. Track is bonded by standard flexible track bonds with mechanical connections. Two hundred and forty-volt direct current is used on two levels and 125-volt current on a third level.

Where ore is delivered to cars through chutes on main haulageways, laybys long enough to hold a trip of cars are provided. In drifts or crosscuts with one draw point only, the cars are spotted in the dead end of the drift by the motor. They are dropped down to the chute, loaded, and passed on to a layby on the haulage side of the chute, where they await the motor. The purpose of this is to minimize hand-tramming. Great care is used in the installation of these features to make easy tramming.

A layby long enough to be a gathering place for cars from all its loading tracks is

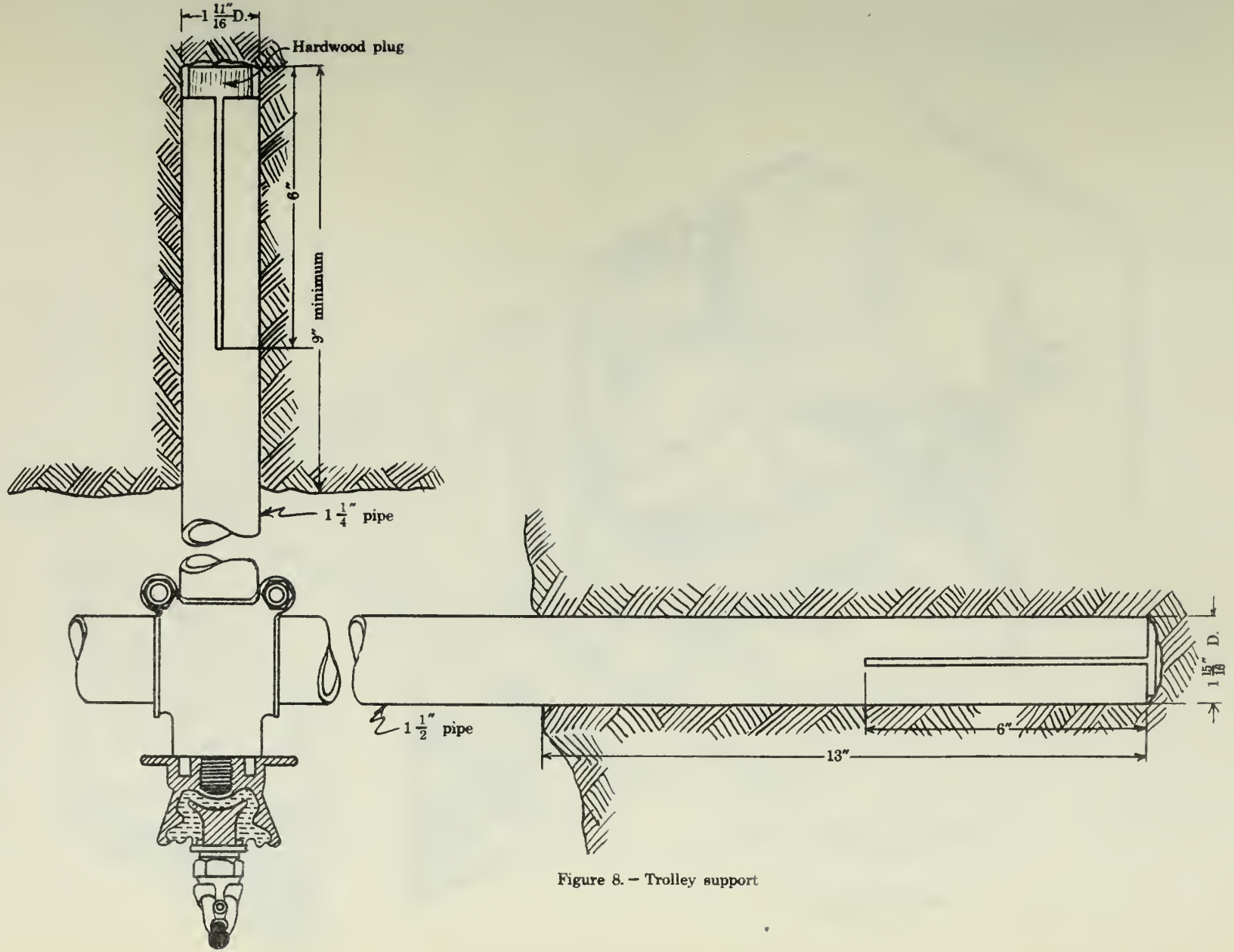


Figure 8. — Trolley support

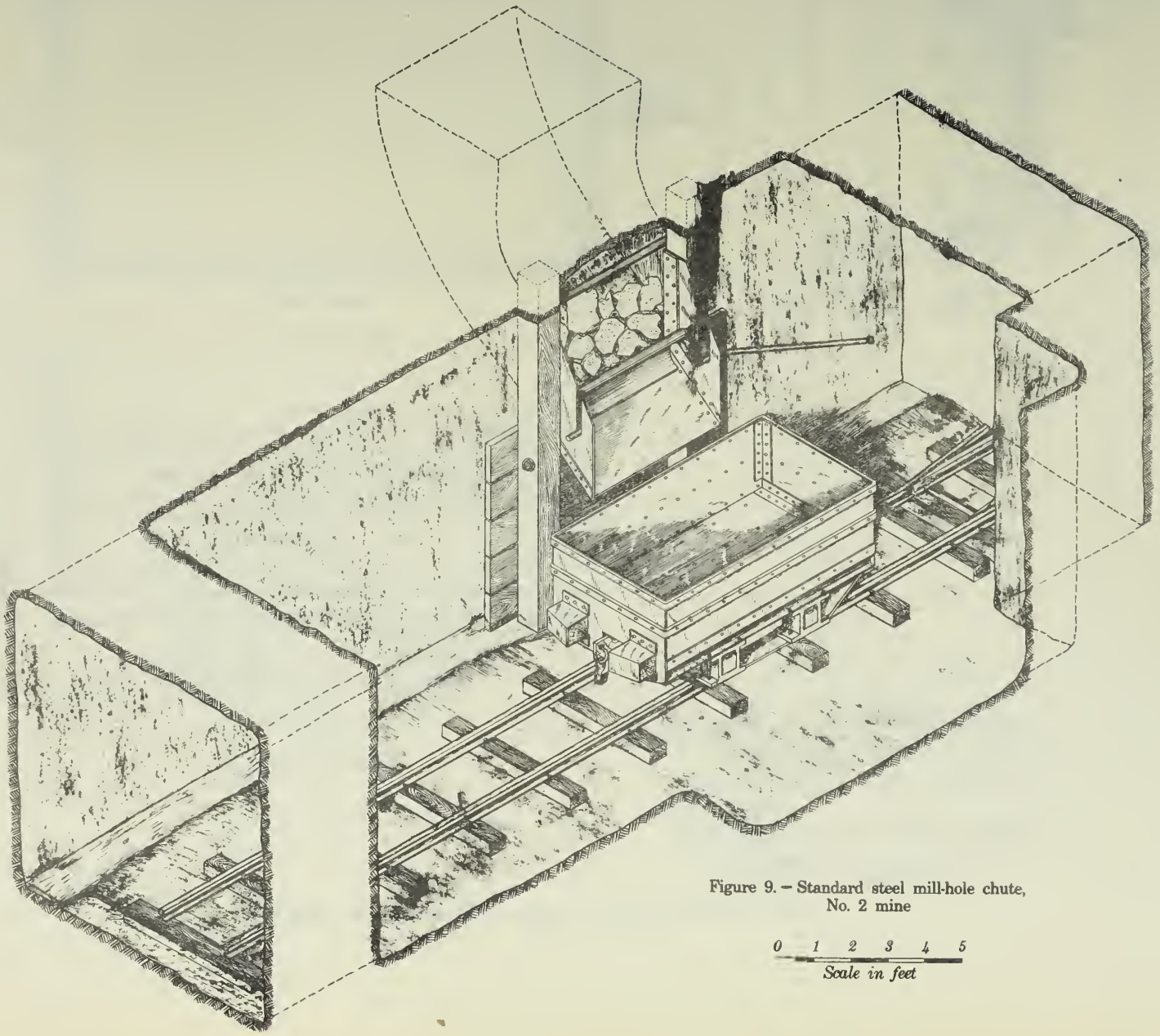


Figure 9. – Standard steel mill-hole chute, No. 2 mine

0 1 2 3 4 5

Scale in feet



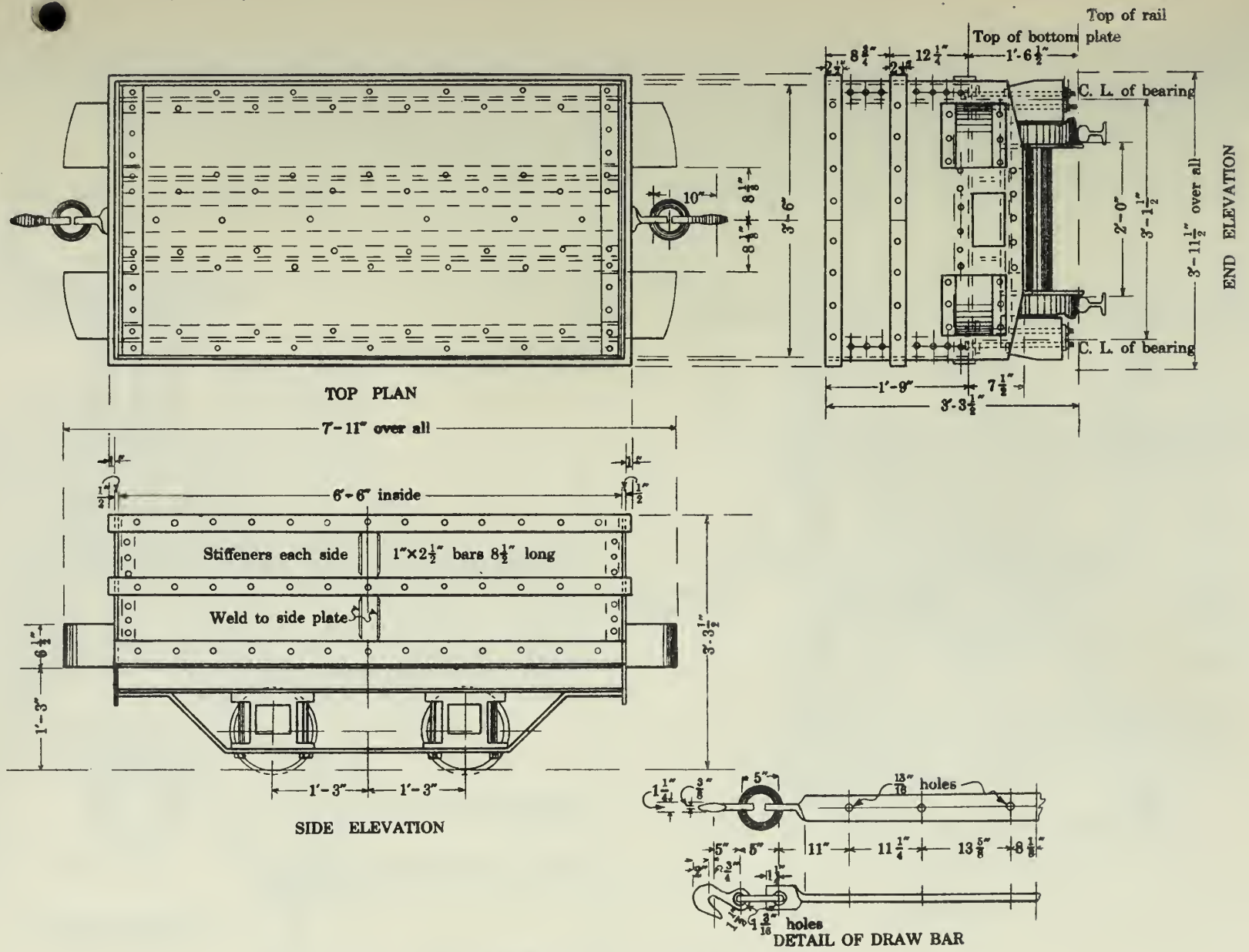


Figure 10. – Two-ton all-steel mine car with railroad-type journal box



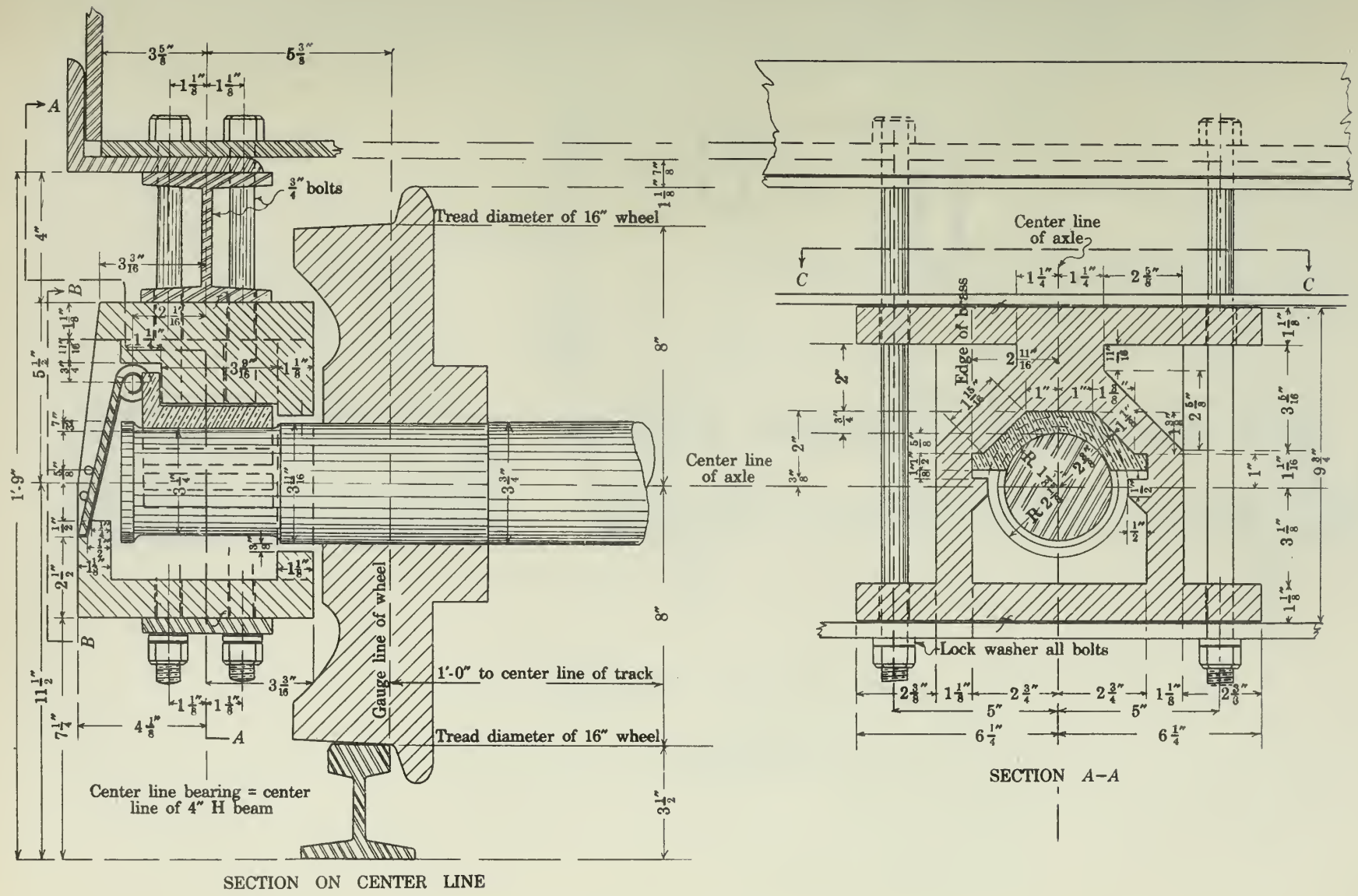


Figure 11. – Journal box and bearing for mine cars

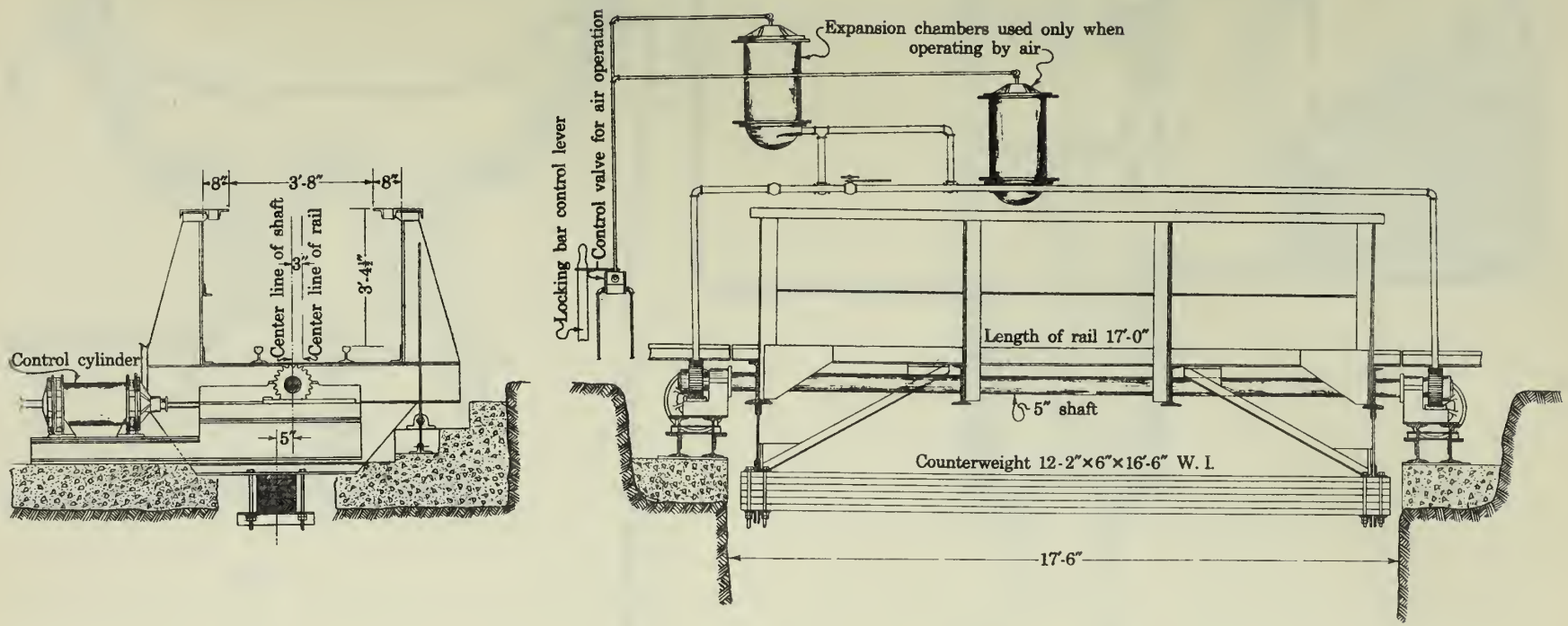
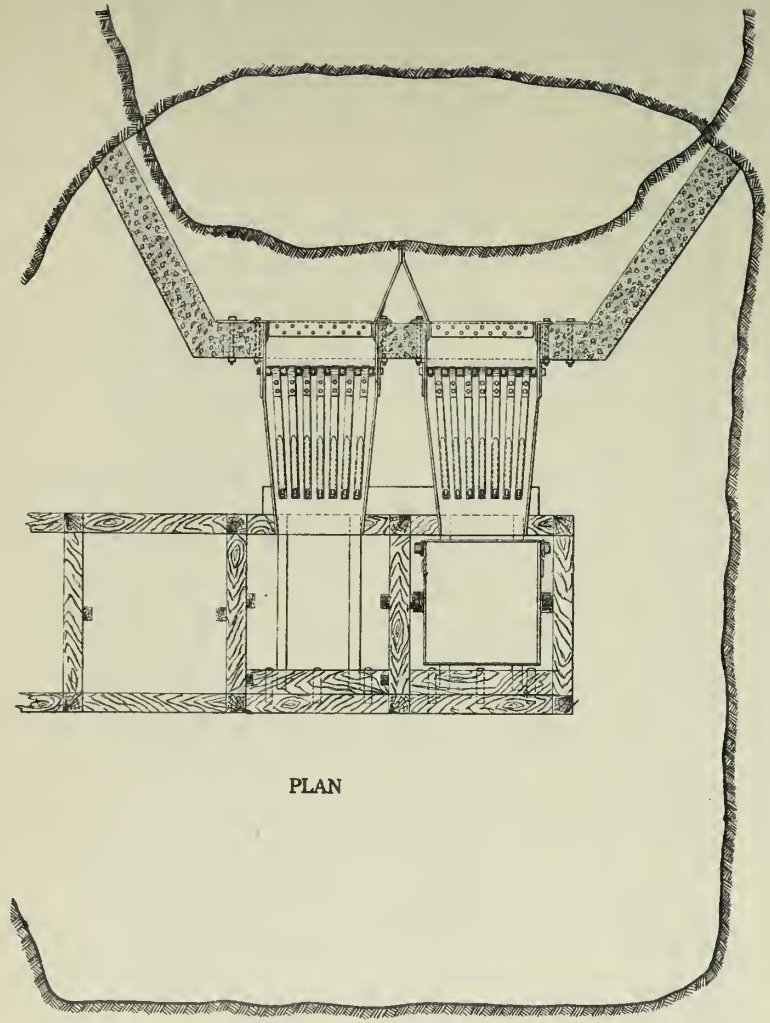


Figure 12. – Two-car shaft-type rotary car dumper

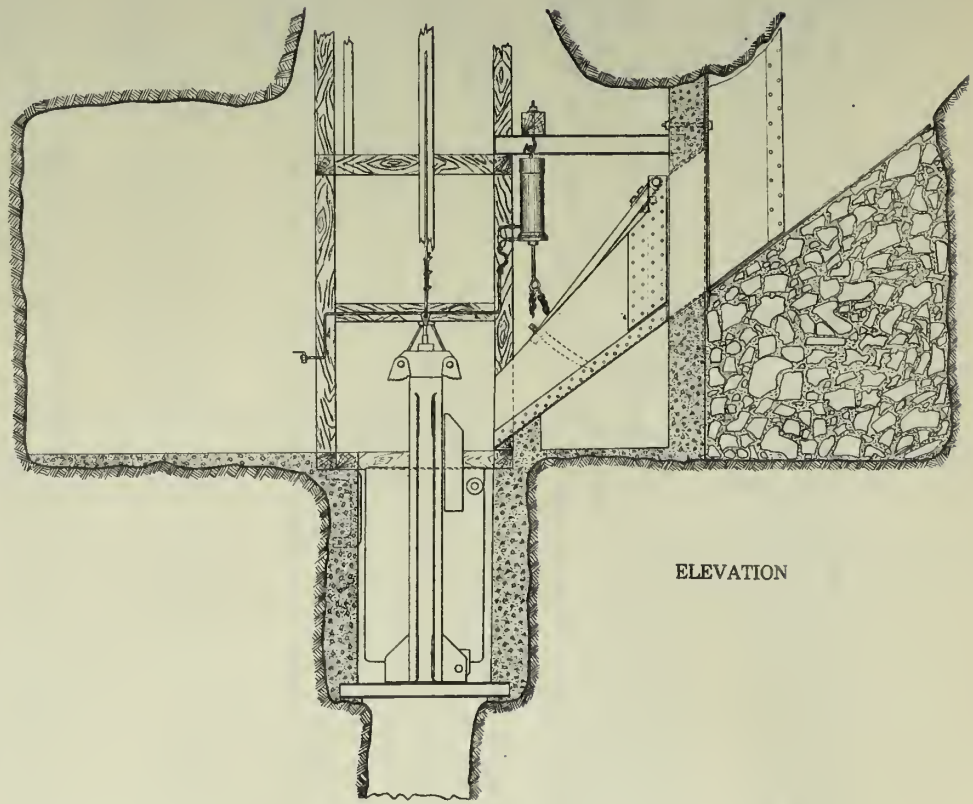
1 0 1 2 3 4 5

Scale in feet





PLAN



ELEVATION

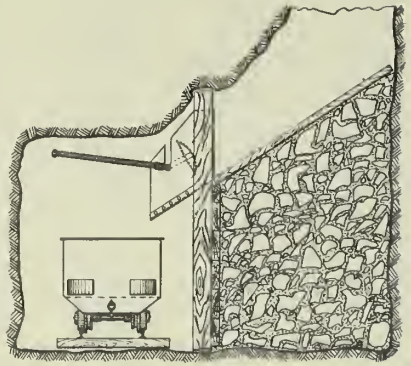


Figure 13. – Skip-loading arrangement, No. 2 mine

0 1 2 3 4 5 10

Scale in feet



installed in each open stope. Cars are assembled on such laybys by mule tramming. Each open stope is furnished one mule. A mule driver is used in gathering cars to laybys in such open stopes served by three or more pairs of shovelers. In stopes served by one pair of shovelers only the shovelers do their own driving.

Mill-hole chutes are of steel. The flow of ore into cars is controlled by a hand-operated gate (see fig. 9). Jackhammers are standard equipment at all chutes and blocking is done at the chute gate. Chute openings are 48 inches wide and 36 inches high. Rocks of a size that will pass these gates are readily handled through tipple and skip loading chutes.

All mine cars are of 2-ton capacity. They are of extra-heavy steel construction, being so built as to withstand the heavy duty incident to chute loading of rough and heavy ore (fig. 10). They are equipped with 16-inch wheels with outside journal-type bearings (fig. 11). In the latter illustration the dimensions given provide for the use of cast iron. If made of cast steel the dimensions indicated by "a" must be used. All dimensions are for finished castings. Steel-incased oak bumper blocks and hook type couplings are used. This type of coupling is believed preferable to other types as regards safety in incline service.

Cars from the lower levels are hoisted to the tipple level in trips of four over a 22° incline. They are transferred to the tipple by a trolley locomotive, which handles cars from all levels through the tipple.

The tipple is of 2-car capacity. It is oil-controlled and so equipped that it may be operated either by air or by gravity (fig. 12). The air-control feature has particular value in holding cars in their dumped position for cleaning.

The tipple delivers into a 750-ton skip pocket, which in turn delivers through chutes into skips. Two skips of 5-ton capacity operate in balance. Skip-pocket chutes are of steel construction. The ore flowage through the chutes into the skips is controlled by air-operated finger-type gates (fig. 13).

WAGES, CONTRACTS, AND BONUSES

Wherever applicable contracting is in effect. Some miscellaneous work of such a nature that it can not very well be contracted for is carried on company time. However, bonuses apply to all company-time work, as will be explained later in this paper.

The mill-hole contractor agrees to break and deliver ore into cars at a stipulated price per ton. The company furnishes drills, drill steel, and power; the contractor furnishes labor and explosives. The chute pullers in turn contract with the mill-hole contractor to draw his ore into cars at a price per ton. Settlement is made with the mill-hole contractor each week, all labor, including his chute pulling, and explosive costs are deducted, and the balance earned on the contract is paid him. An itemized statement of his account is furnished him each week.

Contracts in open stopes cover merely breaking and apply only to tonnage made available to shoveling tracks. Open-stope contractors are required to do all blocking for shovelers and to assist them in keeping their working place in good condition.

The contracts carry no guarantee as to earnings. Modifications of contracts are made when, in the judgment of the management, conditions beyond the contractor's control justify change. As a guarantee of the faithful performance of the contract, a holdback of 30 per cent is retained by the company until a definite amount has been set up. This holdback is payable to the contractor only upon termination of the contract.

Contractors are responsible for secure roof conditions in their working places. If it becomes necessary, through negligence on the part of the contractor, for the company to do this work, deductions are made from this holdback to cover such expense.

Although most contractors pay their drill men and helpers company-scale wages they have the privilege of increasing the compensation received by such men. The company sets a limit to the increase. Any such increased pay becomes a part of the contractor's costs and is deducted on his settlement sheets, hence he tends to make such increases sparingly. The system, however, allows the contractor to give recognization where it is deserved.

Development contracts are based upon footage of advance per week. In most instances a definite distance to be driven is specified in such contracts. Usually these are 2-man contracts, and usually both men are drill men. Settlements are made each week, and their earnings are proportioned to the number of shifts each has worked during the week.

Slushing contracts cover breaking, dragging, and chute pulling. Payments are made on a tonnage basis, as in other contracts. The contractor supplies all labor and explosives and takes care of all blocking, cable splicing, and hoist repairing.

Motor men and motor couplers participate, in proportion to their earnings, in a bonus based upon the total weekly tonnage handled. Skip tenders and helpers participate in a like bonus.

The day-shift mine foremen participate in a monthly bonus based upon the monthly production of 60 per cent zinc concentrates. As the night shift is principally a chute pulling and shoveling shift the night shift foreman's bonus is based upon total tonnage hoisted.

VENTILATION

A 45,000-cubic foot buffalo forge conoidal fan supplies such artificial ventilation as the mine requires. It is located at the collar of No. 4 shaft - a ventilating shaft - about 4,000 feet from No. 2 shaft. The various working places are so interconnected by stopes, raises, etc., as to provide a good natural circulation. The intake and outlet of the fan are so arranged that the ventilating shaft may be made either an upcast or a downcast, as desired.

MINE LIGHTING

The shaft station and haulageways are electrically lighted. Workmen furnish their own lamps, but the company supplies carbide.

DRAINAGE

Mine drainage is by open ditches paralleling haulageways. On No. 1 level the drainage is direct to the main sump and pump station at the shaft. Water on the lower levels is ditched likewise to secondary sumps, from which it is either relayed to the main-level sump or into flood basins, whose purposes is to store excess water during particularly wet periods, or in case of power failure. Bulkheads of concrete and steel construction in various drifts and crosscuts throughout the mine supply added protection against heavy inflows of water.

SAFETY AND WELFARE

Safety and welfare work begins with the physical examination of all applicants for employment. Safety propaganda is forwarded by bulletins and display posters. Accident statistics, compensation, insurance, rental and repairs, sanitation, employment, welfare, police, and camp maintenance are grouped under the supervision of the personnel director.

An emergency hospital is maintained, and a doctor and visiting nurse are in regular attendence. First-aid cabinets, stretchers, blankets, etc., are provided in all departments. Minor first-aid treatments are administered in the various departments by men holding United States Bureau of Mines first-aid certificates.

An applicant for work is interviewed at the employment office to ascertain his fitness. He must pass a physical examination. Record is made of his previous employment. Before going to his working place in the mine he is instructed in safety matters by a safety instructor, for whom an office underground is furnished for this use.

Foremen's meetings are held monthly (See fig. 14). The chairman of the general safety committee presides at these meetings, the purpose of which is to encourage open discussions of safety work. Accident records are presented and safe and unsafe practices are discussed. These and all other safety meetings are held during working hours and attendence is a part of a foreman's duties.

A dinner is given the foremen by the management whenever the number of lost-time accidents during a quarter is less than during the corresponding quarter of the preceding year. Some form of entertainment is presented in connection with these dinners. Occasionally foremen are called upon for remarks.

Monthly bonuses are paid to foremen with the best safety records. A foreman's record for this purpose is based upon his man-hour exposure.

At the end of the year a bonus is given to each man employed underground who has worked throughout the entire year (time of vacation and sickness excepted) without a lost-time accident. A like bonus is given to surface men.

All lost-time accidents are investigated by a committee composed of two members of the general safety committee and the foreman in whose department the injury occurred. These investigations have a threefold purpose: 1, To establish the real cause of the accident (the place where the accident occurred is inspected and witnesses are questioned); 2, to establish the responsibility of the accident; and 3, to make suggestions that will help to avoid similar accidents. The results of these investigations in detail are posted on all bulletin boards at the plant.

Periodically, first-aid teams are organized in each department. In this way plant competition is encouraged. All first-aid training is done on company time. Instructors are company men holding United States Bureau of Mines first-aid certificates. Their work is checked up by U. S. Bureau field men from time to time.

Bulletin boards, film slides, and weekly publications are used to bring safety matters to the attention of employees.

COST DATA

Mascot No. 2 mine
Dec. 27 - 1928 - Oct. 30 - 1929

Tons hoisted:	Total tons	Percent of total
Hand loading (contract)	89,177	16.87
Hand loading (co.acct.)	15,936	3.01
Mill holes (contract)	397,589	75.21
Mill holes (co.acct.)	8,681	1.64
Development (contract)	16,800	3.18
Development (co.acct.)	508	.10
Skip pocket and bin adjustment	- 65	- .01
Total	528,626	100.00

Summary of Costs per Ton

	Labor	Super-vision	Compressed air, drills, and steel	Electric power	Explo-sives	Other supplies	Total
Development	$0.028	--	$0.004	--	$0.015	$0.006	$0.053
Mining	.130	--	.042	--	.045	.005	.222
Transportation	.221	--	.001	$0.014	--	.042	.278
General	.037	$0.038	--	.002	--	.009	.086
Total	.416	.038	.047	.016	.060	.062	.639

Costs in Units of Labor, Power and Supplies

Labor (Man hrs. per ton	Mining - only
Drilling and blasting	0.214
Timbering	.009
Mucking	.093
Chute pulling - haulage and hoisting	.368
Supervision	.027
General	.120
Total labor underground	.831
Average tons per man shift (9 hrs.)	10.83
Labor, per cent of total cost (inc. dev.)	71.05

Explosives and power

Explosives, kind	30 per cent low-freezing gelatin
Explosives, lbs. per short ton	.502

Power, kw. h. per short ton	Mining and pumping
Compressed air	3.81
Noisting	1.06
Haulage and lighting	.63
Pumping	4.04
Ventilation	.05
Mechanical loading (slushing)	.09
Total, kw. h. per ton hoisted	9.68

Development-cost data

Feet advanced from Dec. 27, 1928, to Oct. 30, 1929

	Feet
Drifts and crosscuts	2,015
Raising	538
Raise reaming	628
Layby slabbing	193
Total	3,374

	Per foot advance	Per cent of total
Labor	$4.07	49.77
Insurance	.28	3.48
Explosives	2.25	27.48
Material	1.00	12.23
Power	.57	7.04
Total	$8.17	100.00

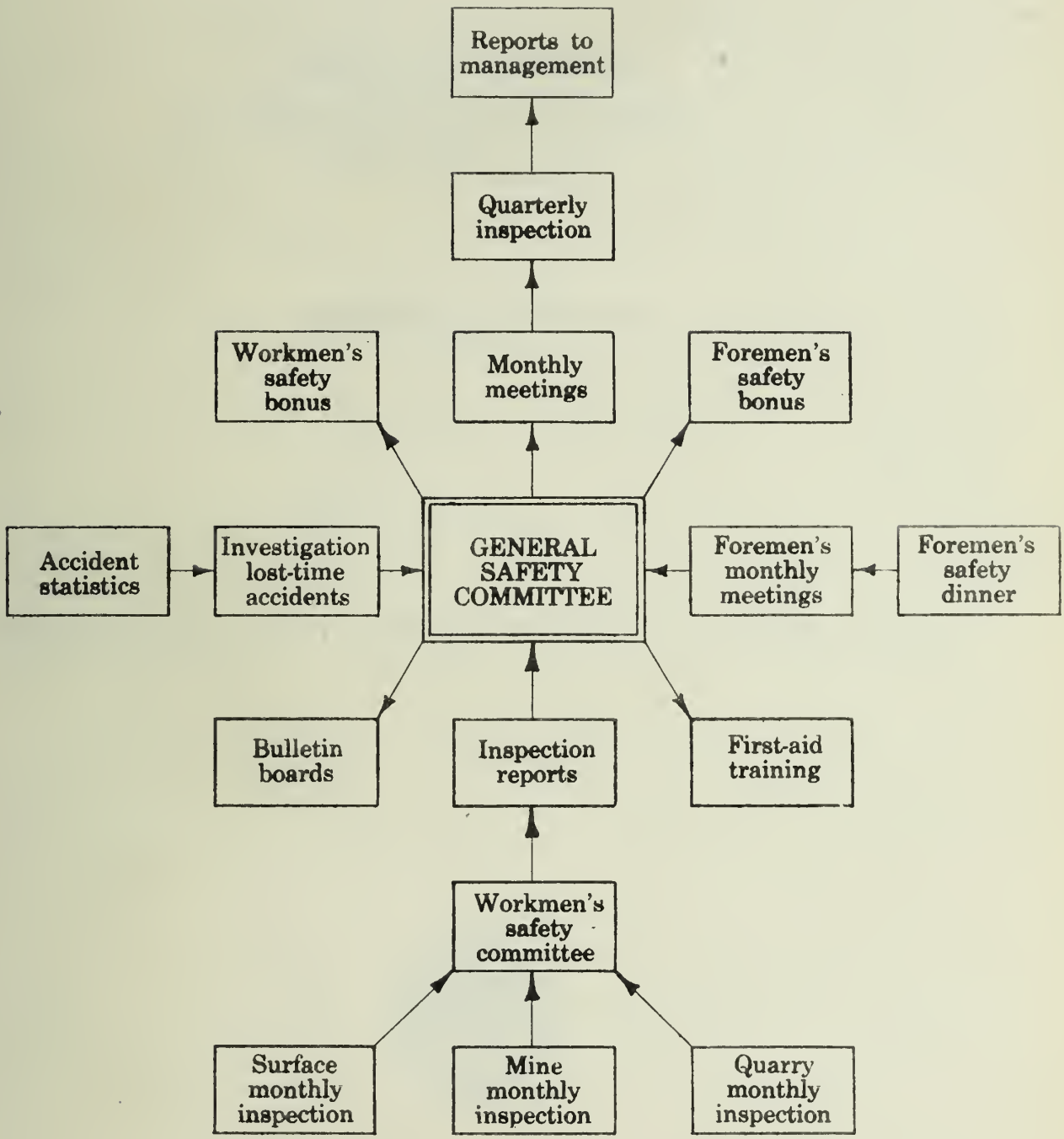


Figure 14. — Safety organization chart



I. C. 6240 March, 1930

DEPARTMENT OF COMMERCE
UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

MINING PRACTICE AT HARMONY MINES CO., BAKER, IDAHO.

BY

R. DUNCAN GARDNER

I.C.6240.
March, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE – BUREAU OF MINES

MINING PRACTICE AT HARMONY MINES CO., BAKER, IDAHO[1]

By R. Duncan Gardner[2]

INTRODUCTION

This paper describing the mining practice at the Harmony mine, Baker, Idaho, is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts in the United States.

About 150 tons of copper ore containing 4 per cent copper, 0.001 ounce gold, and 0.1 ounce silver, is being produced daily from the Harmony mine by the shrinkage method of mining. The ore is taken by an aerial tramway 1/3 mile long to the mill, where it is concentrated. The concentrate is shipped 8 miles by truck to the railroad and thence by rail to Utah for smelting.

LOCATION AND HISTORY

The mine is in the Lemhi Mountain Range, 18 miles southwest of Salmon, Idaho. Ore was first discovered about 1870, during the period when an intensive search was being made throughout the West for the mother lodes of the great placer fields that were then being worked. For may years mining was intermittent, and primitive methods only were employed. It was not until 1918 that operations were begun on a scale commensurate with the size of the ore body.

GEOLOGY

The ore occurs in sheer zones in thick beds of highly siliceous shale. Outcrops of the principal vein have been traced for 12 miles. No igneous rocks have been found close to the ore bodies, but some basic porphyry dykes and a great granite batholith in the district have undoubtedly affected the fracturing and mineralizing of the zones under consideration.

The ore occurs in regular shoots or pipes of primary chalcopyrite which has been

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1 – The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6240."

2 – One of the consulting engineers, U. S. Bureau of Mines.

partly altered in the upper levels to bornite and chalcocite. The shoots of ore are included within the fractured zones and are uniform in their principal characteristics. The strike of the ore bodies is N. 40°W. and the dip is 60 to 65° SE. The main ore shoots average 5 feet in thickness and 80 feet in length. The gangue mineral in the vein is quartz and altered country rock. Figure 1 is a cross section of the main ore shoot.

PHYSICAL CHARACTERISTICS OF ORE AND INCLOSING ROCKS

The vein filling and ore are hard and difficult to break. Planes of weakness that occur in the vein parallel to the walls assist the explosives in breaking the ore.

The hanging and foot walls of all stopes are hard shale which stands up very well and shows no appreciable alteration after years of exposure.

The physical characteristics of the wall rocks and ore are very favorable for the use of the shrinkage method of stoping, which was chosen for its simplicity and saving in labor and supplies.

METHODS OF PROSPECTING AND EXPLORATION

The regularity of occurrence of the ore bodies has made only a small amount of prospecting necessary and has involved only drifting along the vein through the various barren areas that divide the ore shoots within the single zone worked to date. Some post-mineral faulting has occurred, and some prospecting was necessary to find the downward extension of the ore from the 300 level.

Figure 1 illustrates conditions in the main ore shoot when the ore had been mined from the 800 level to the fault zone. The direction of the throw of the fault was easily ascertained, so comparatively little prospecting was necessary to locate the ore above the fault.

A few short crosscuts run beyond the walls of the ore zone have yielded little additional ore.

METHODS OF DEVELOPING AND MINING

The mine is opened by four tunnels 100, 200, 300, and 800 feet vertically below the outcrop. Tunneling was chosen in preference to sinking shafts in developing the mine because of the rugged condition of the mountain slopes and the presence of water in the ore zones. Tunnels are used for both haulage and drainage. The last tunnel to be driven is 2,000 feet long and cuts the ore body about 800 feet below the outcrop.

From the point where the tunnel cuts the ore 6 by 8 foot haulage drifts are run along the vein to the extremities of the ore shoot. Recently an inclined 6 by 8 foot winze was sunk 150 feet on the main ore body. From the bottom of this winze a sublevel was developed similar to that on the main level, except that the drifts are smaller in cross section (4 1/2 by 6 1/2 feet). Stopes on this level have just come into full production. The mining cost here will determine the advisability of sinking to a still deeper level.

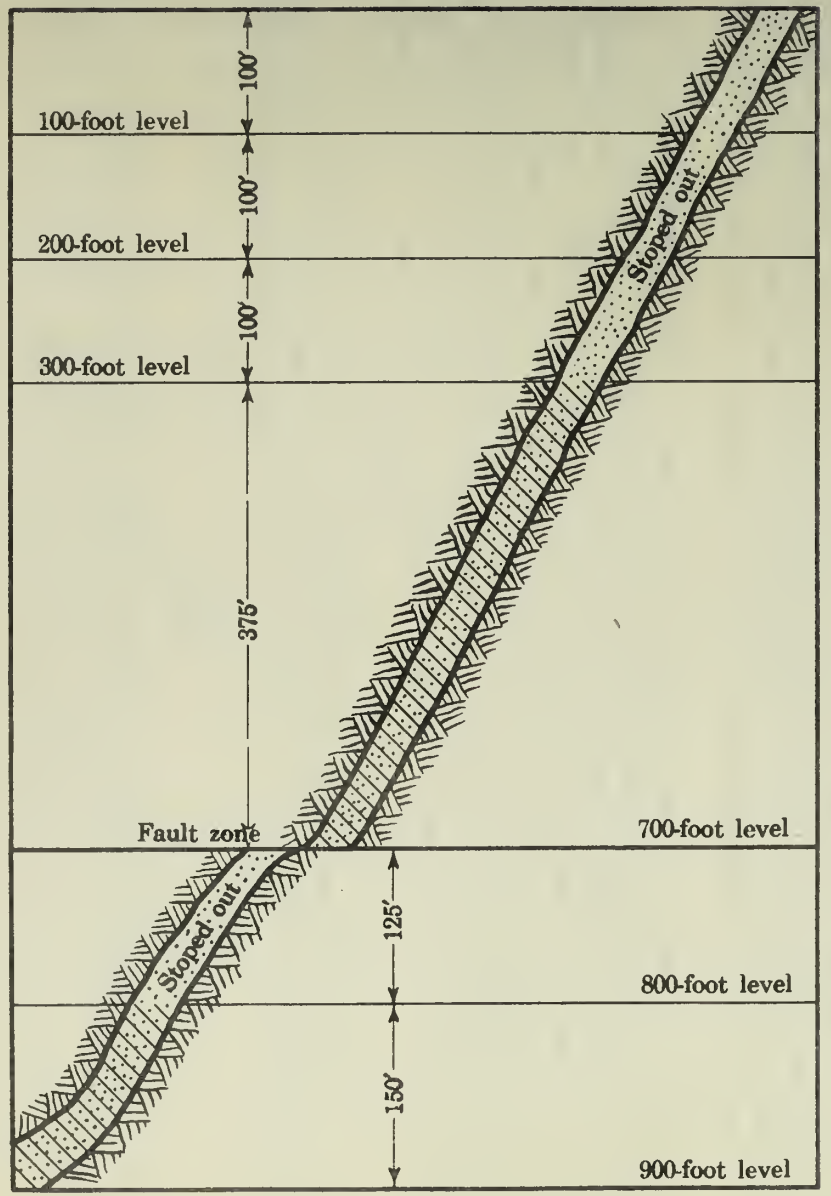


Figure 1. – Cross section 900-foot level to outcrop

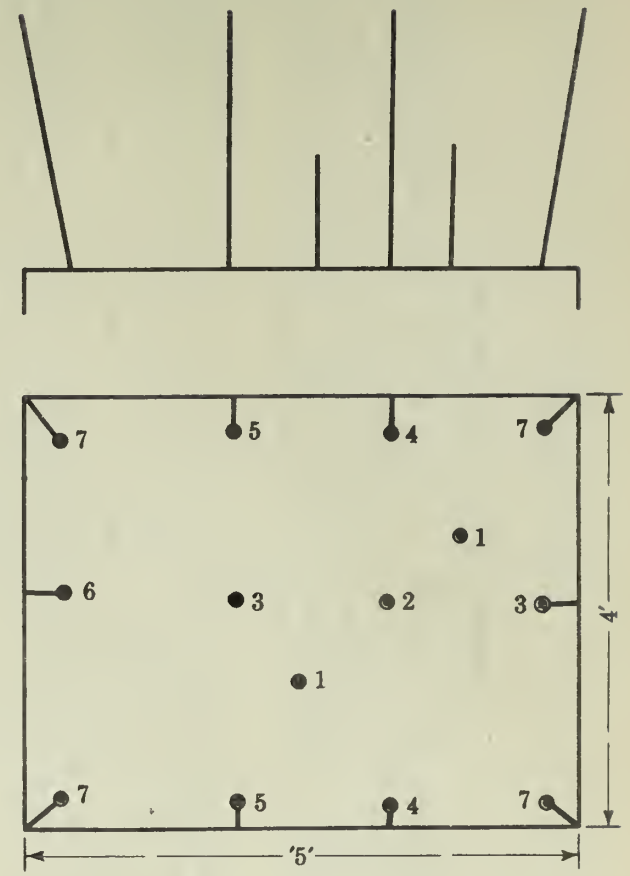


Figure 2. – Raise round, numbers indicate order of firing



DEVELOPMENT DETAILS

Tunnels, Drifts, and Crosscuts

The main haulage tunnel is 6 by 8 feet in the clear. The first 300 feet from the portal required timber support. This necessitated excavating to a cross section of 7 1/2 by 9 feet. Tunnel sets supporting heavy ground were made of 8-inch round timber and consisted of two posts, cap, and sill. Posts were cut 7-1/2 feet long. Round lagging 3 inches in diameter was placed behind the posts and above the cap.

All drifts and crosscuts are driven with medium-weight leyner-type drifters using 1 1/2-inch hollow-round steel with lug shank and standard double-taper cross bit. Three changes of steel are used for drilling drift rounds. The gauge of the starters is 2-5/8 inches, the second steel 2-1/2 inches, and the finisher 2-3/8 inches. Drilling machines are used either on a column and cross arm or a crossbar. The drifting crew consists of three men. The machine man and a helper clean up any broken material left from the previous round, drill and shoot the round, and keep the track in shape. A shoveler on the opposite shift is usually kept busy mucking during the entire period.

Practically all tunnel work and drifting are done by contract. The contractors are paid so much a foot and the company furnishes machines, tools, steel, and explosives. The contractor has the privilege of using any standard round that he desires, but with few exceptions the ordinary round with downward-cut holes is used in all drifts and crosscuts. Rounds are usually drilled 5 1/2 feet deep and contain from 17 to 22 holes. An average of 4 feet is usually broken, but it is not exceptional to find a crew that will consistently pull 4 1/2 to 5 feet after the members have learned the characteristics of the ground.

Fifty pounds of 40 per cent strength gelatin dynamite is used for blasting each drift round. Some experimenting has been done with stronger explosives, but the results obtained did not warrant their use. When 60 per cent strength explosive was used exclusively the rounds were broken better than with 40 per cent gelatin dynamite, but formation of an excessive amount of gas and a too great concussion after each shot made the use of the higher-grade explosive unsatisfactory. The use of the 60 per cent strength explosive in the cut holes or in replacing a few cartridges of the 40 per cent in the bottom of all the holes gave a slight improvement in breaking, but everything considered the use of the higher explosive was not considered advantageous.

To insure the drilling of 20 holes, 5-1/2 feet deep, each shift air must be supplied to the drills at a pressure in excess of 90 pounds. Each crew is furnished 20 sets of steel at the beginning of the shift.

Raises

Raises are of two sizes - 5 feet by 12 feet to contain a double chute and 5 by 7 feet for a single chute or manway. Occasionally a small raise 4 by 4 feet or 4 by 5 feet is driven to prospect the hanging wall or to make a connection.

In all raises the same style of round is used. This round is a departure from the usual practice but works very well in every kind of rock in which it has been tried. Its superiority over the ordinary round is marked in exceptionally hard ground. It consists of the usual number of holes drilled nearly parallel to each other straight into the face. In addition to these, two short holes one-half the length of the other holes are drilled diagonally across the middle holes of the round. Figure 2 shows the position of the holes for a 5 1/2-foot round. The short holes break well and form a cavity in the center of the face. The rest of the round then has a fair chance of breaking to this center cavity. A 3 by 4 foot raise has been driven through very hard limestone and advanced an average of 5 feet per round when this style of round is used. In such a small raise much pitch can not be given to cut holes; but by drilling seven long holes and two short ones no difficulty was experienced in putting in the rounds, and each broke to the depth of the holes drilled.

Raise crews consist of a miner and a helper.

STOPING METHOD

After the haulage drifts have been driven to the limits of an ore body the stope is developed for mining by installing double chutes on 40-foot centers and manways at 120-foot intervals. If the stope is under 80 feet long only one manway is maintained. Raises for the chutes are run 5 by 12 feet in size, 18 feet above the back of the drift, by a regular raise crew, which is then transferred to another raise. A timber crew then constructs the chutes and a manway, then the sides of the raises are stoped out, as shown in Figure 3, and grizzlies installed. As soon as the chutes are built a 4 by 5 foot raise is put up at the far end of the ore body. The raise parallels the pitch of the ore body and is at the end of the shoot. The raise is primarily run for ventilation but is also the starting point of stoping rounds.

The purpose of mining operations at this point is to leave a pillar above the chute to form a protected chamber for bulldozing and expediting the flow of ore through the chutes. To accomplish this with least effort the crew drives a large inclined raise from each side of the chute. These raises are driven at an angle of 40 per cent with the horizontal. Approximately 12 feet above the chute nearly flat raises are driven from the back of these inclined raises with the object of meeting above the pillar, as illustrated in Figure 4.

Pairs of flat raises are also extended from adjoining chutes to meet each other, leaving a pillar above the drift between the chutes. The pillar is about 14 feet long and 8 feet high at the middle. One pillar is cut in two for the manway in the stope, as shown in Figure 3. The manways are cribbed through the broken ore with round 8-inch timber and have two compartments, one 2 1/2 feet by 4 feet for a ladderway and one 3 feet by 4 feet for a slide for getting supplies into the stope.

Mining proceeds along the entire face of the stope, as soon as the work around the pillars is completed.

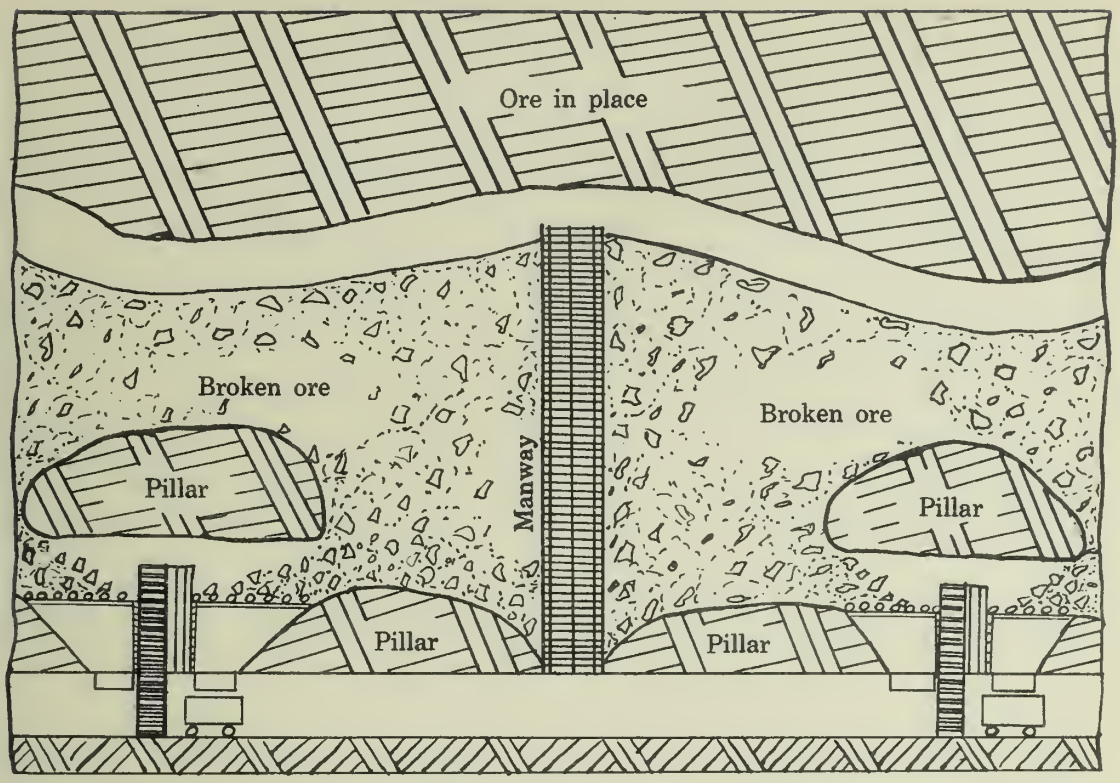


Figure 3. – Longitudinal section through stope showing shrinkage method of mining

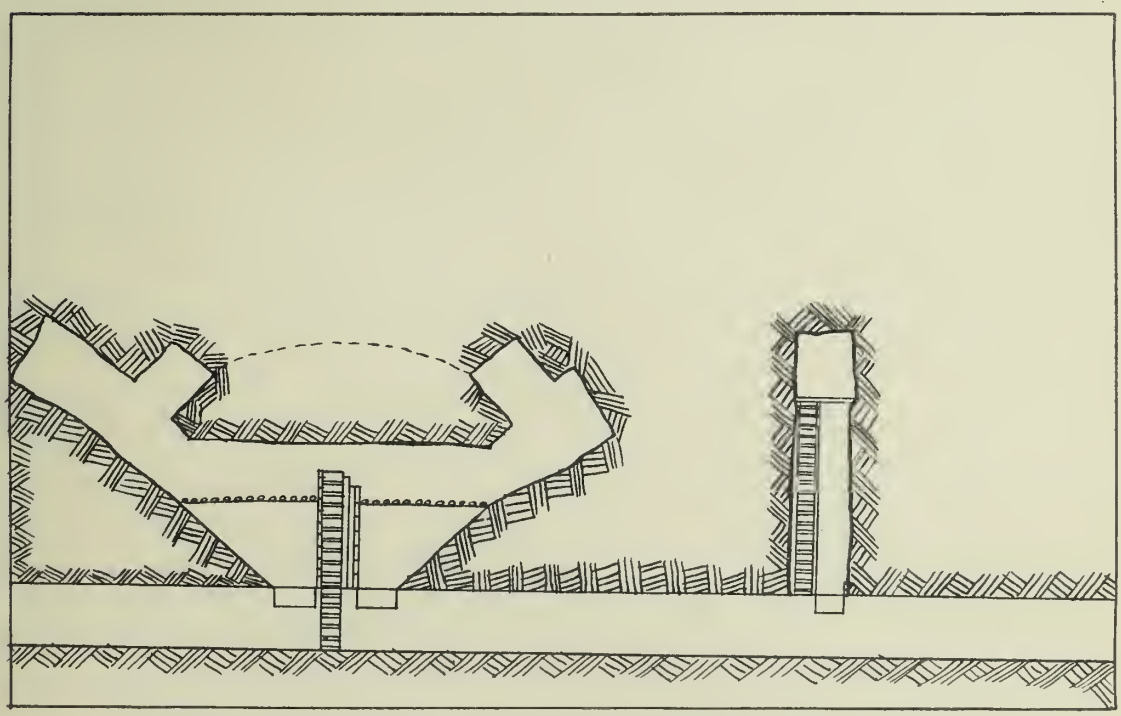
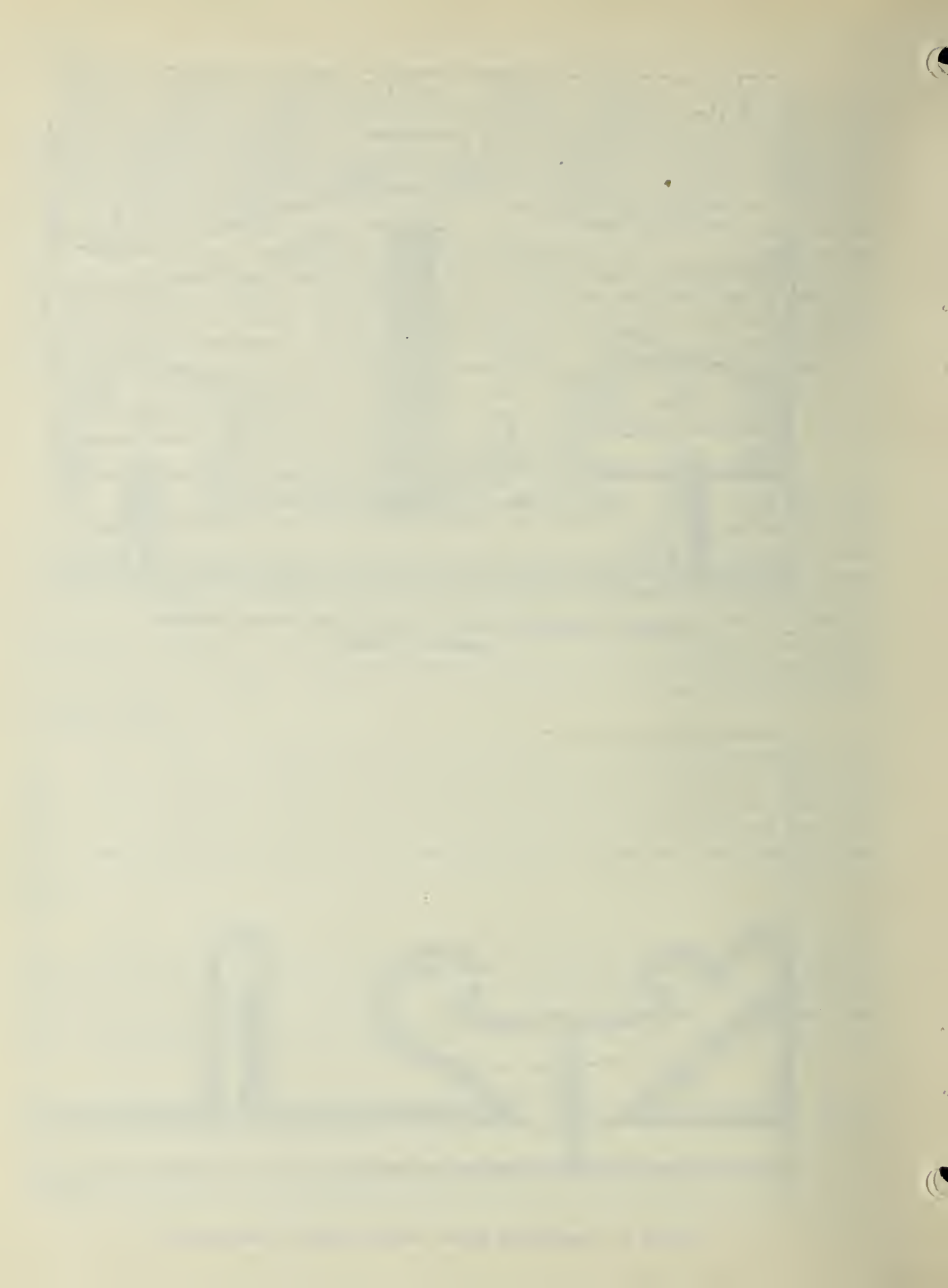


Figure 4. – Longitudinal section showing method of starting stopes



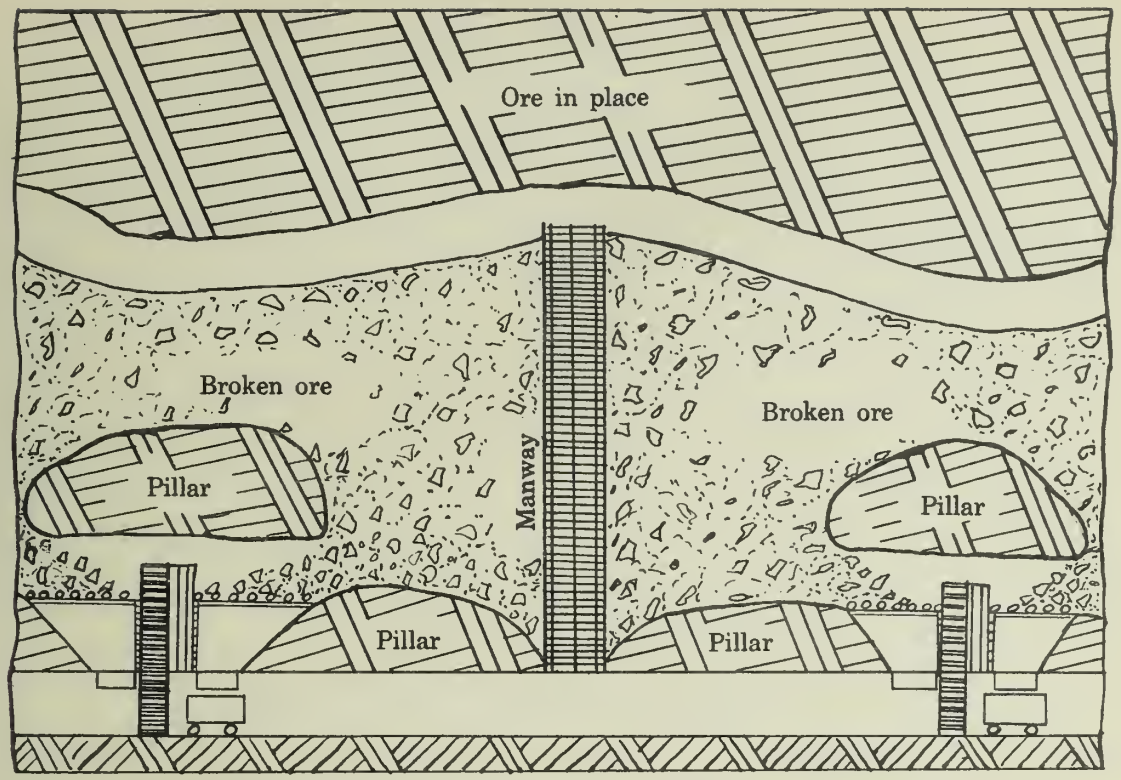


Figure 3. – Longitudinal section through stope showing shrinkage method of mining

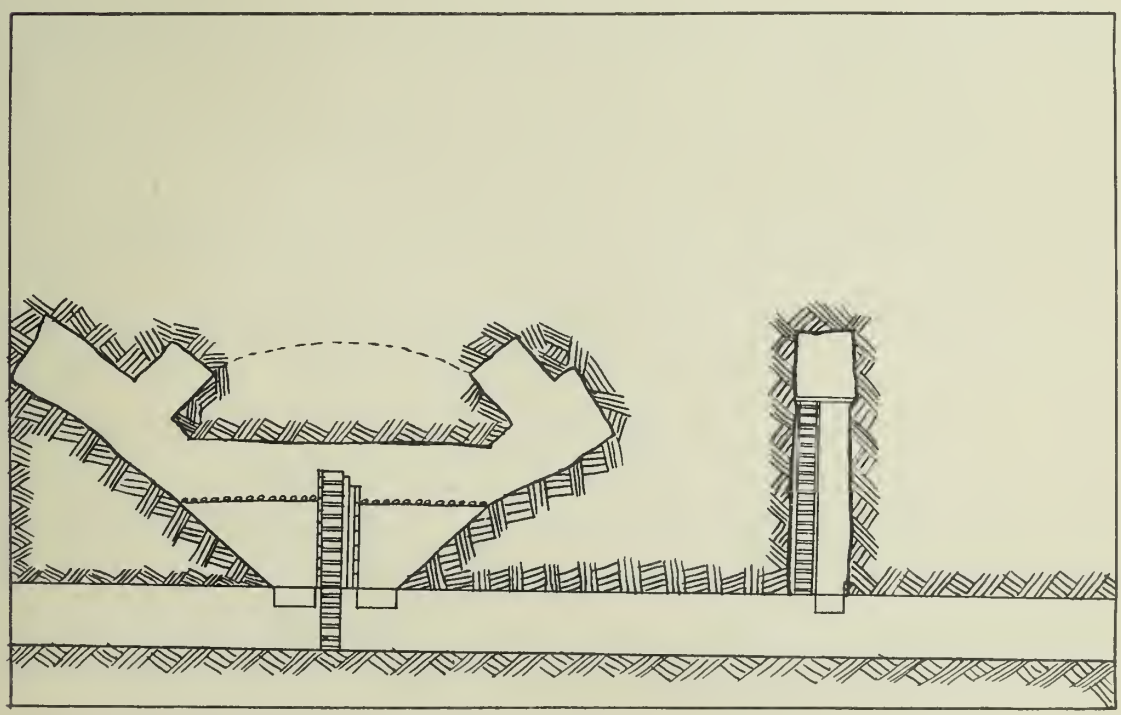


Figure 4. – Longitudinal section showing method of starting stopes





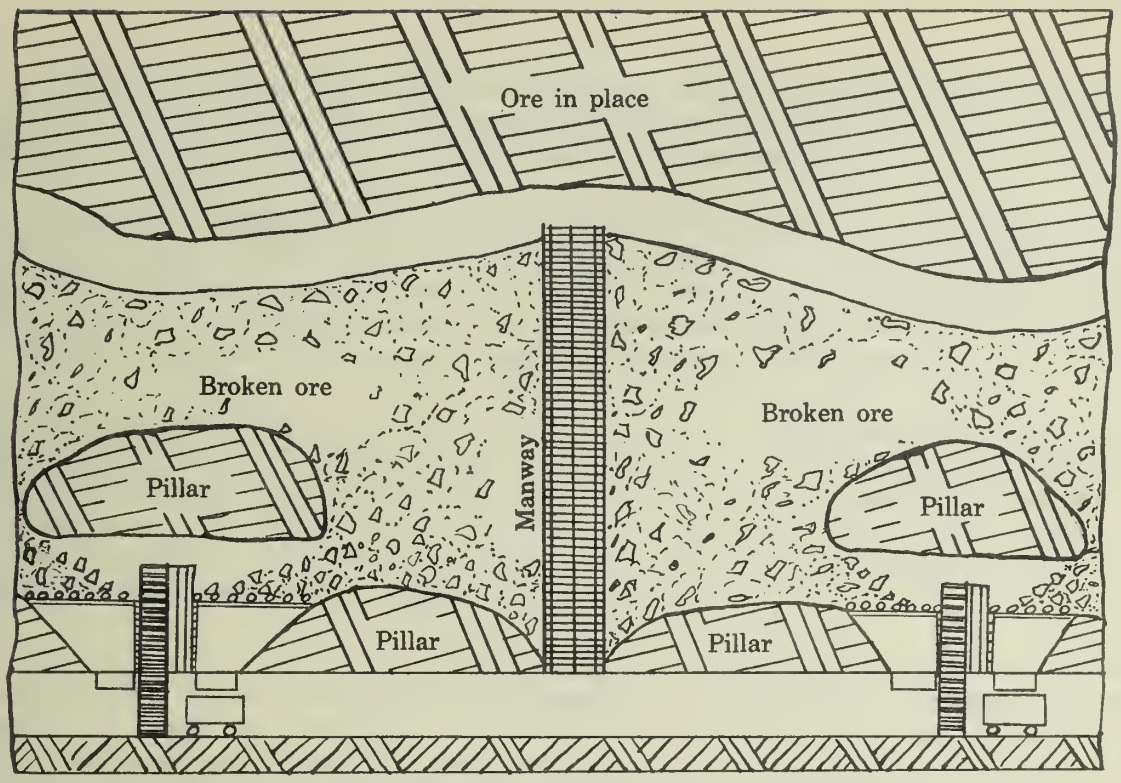


Figure 3. – Longitudinal section through stope showing shrinkage method of mining

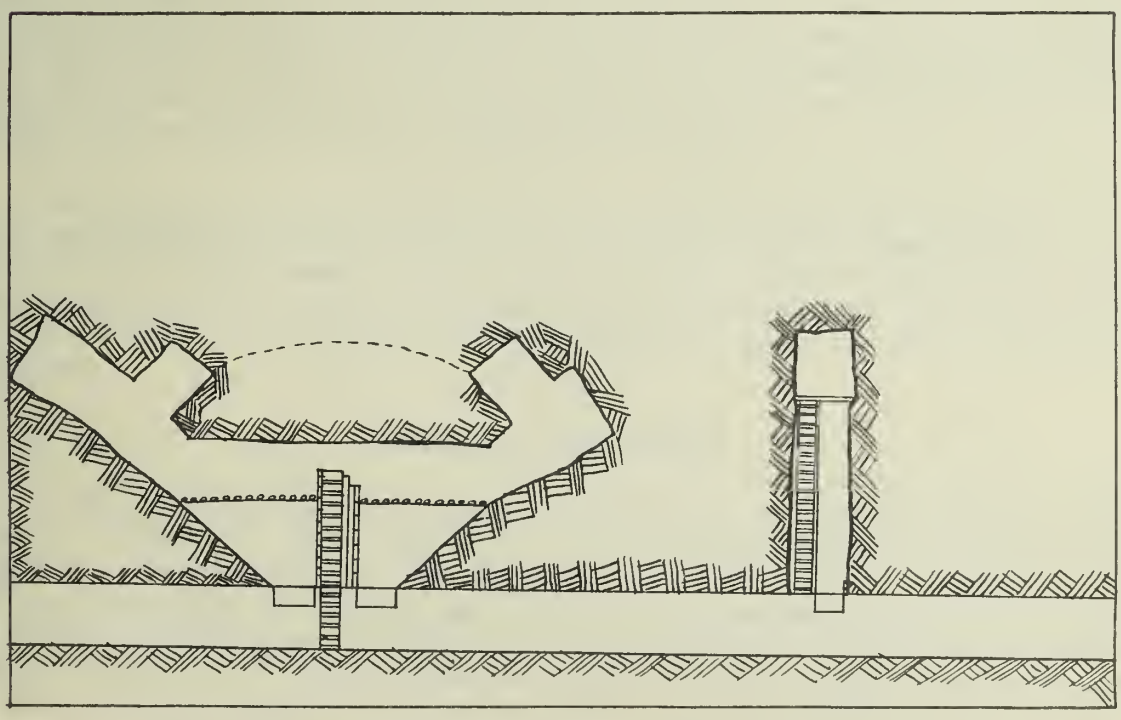


Figure 4. – Longitudinal section showing method of starting stopes





As the breaking of ore in the vein proceeds, enough is pulled from the chutes to keep the broken ore at a convenient distance from the back of the stope. Approximately 40 per cent of the total ore broken is pulled during stoping operations.

Stoping is continued beyond the level above until all pillars are broken. Pillars left to support a heavy hanging wall or because the ore was too low a grade to mine are not taken out; such instances, however, are rare. As soon as breaking is completed the broken ore is pulled from the chutes as needed. On rare occasions it has been found necessary to place a timber bulkhead in order to avoid waste caving on the broken ore.

During the period of operations when ore is being broken around the pillars above the chute a small amount of hand-shoveling is necessary, but after stoping has proceeded above the pillars no more shoveling is required. The drilling crew must level off the top of the broken ore before beginning to drill; but as a rule the ore lowers uniformly as the chutes are drawn, so little labor is required for putting a stope in shape for drilling. No blockholing is necessary in the stopes.

Drilling in all stopes and raises is done with automatically-rotated wet stopers. Formerly 1 1/4-inch hollow round steel with a standard lug was used, but this has been found too heavy, so a change has been made to 1-inch quarter octagon. This change has had the effect of increasing the efficiency of both miner and machine. Three lengths of stoper steel are used - the gauges being 2 5/8, 2 1/2, and 2 3/8 inches, respectively.

The crew starts drilling at one end of the stope and advances toward the manway. Stope rounds break to the raise at one end of the stope; raise rounds are drilled in the other end as each cut is started. Holes drilled each shift are shot at the end of the shift. The manway is cribbed above the broken ore, and new cribbing is added just before shooting of the nearest holes. As soon as the entire cut has been drilled and shot the ore is pulled from the chutes to make room for another round. About the right amount must be pulled from the stope before the crew enters to resume drilling. For a 6-foot vein this amounts to about 1-1/2 tons for each linear foot of stope above the chute.

When the chute is being pulled one man is needed in the bulldozing chamber. He works the ore through the grizzlies and drills and shoots any bowlders that are too large to be broken readily with a hammer. For this work a light jackhammer and 7/8-inch hollow hexagonal steel are used. Sixty-pound rails are used for grizzlies and are set 18 inches apart. The ore breaks in large pieces, and considerable secondary breaking is necessary at the grizzlies.

UNDERGROUND TRANSPORTATION

A 10-ton storage-battery locomotive handles all ore through the main haulage tunnel for 2,000 feet. It pulls a train of 8 to 10 side-dump cars of 2 1/2-ton capacity each trip. All ore is pulled during the day shift, and the batteries are charged at night. The pulling of the entire day's run of ore in one shift is necessary in the winter because wet ore freezes in the bins at the head of the aerial tramway. To avoid this during the cold weather practically no ore is allowed to remain in the bins longer than a few hours. Care is taken that all of the bins are emptied before the tram is stopped at the end of the day's shift.

Thirty-pound rails are used in the main haulage level and 12-pound rails in sublevels. Ties are untreated pine, 4 by 6 inches in cross section and 3 feet 6 inches long. The grade of the main level is 0.5 per cent. The gauge of all tracks is 18 inches. Hand-tramming on sublevels makes a smaller car necessary. For this purpose 16-cubic foot cars are used.

Hoisting through the winze below the 800 level is done with an electric hoist. This hoist has a rope pull of 3 tons and a speed of 200 feet per minute when loaded to capacity. It is operated by a 440-volt, 50-hp., alternating-current motor equipped with a variable switch control starter. An especially designed skip of 36-cubic foot capacity is used to hoist the ore from the sublevels to the storage pocket above the main tunnel.

PERCENTAGE OF EXTRACTION

Virtually all ore of commercial grade is extracted. Occasional small pillars consisting of ore of too low a grade to handle are left. As the pillars above a level are reached and extracted by stoping from a lower level, very little ore is lost.

VENTILATION

A suction fan at the portal of the tunnel is run continuously and by means of a 10-inch ventilator-pipe system draws smoke and gas from the various workings. Connections between the various levels keep them practically free from gas.

Air and gases usually move upward through the ventilation raises at the end of the ore shoots, and no trouble is experienced in keeping the stopes free from gas. Occasionally the draft in a raise will be downward, due to water dripping through abandoned stopes above. When such is the case a branch of the ventilation suction pipe is run up the manway furthest from the ventilation raise and quickly frees the stope of gases from blasting. Virtually no gas reaches the haulage level from the stope. The small amount of gas produced when bowlders are blockholed in the bulldozing chambers is usually sucked into the ventilation pipes within 10 to 15 minutes after the charges are fired.

Temperatures throughout the mine are relatively low, ranging between 50 and 60°F. at most of the working faces. The main tunnel is uncomfortably cool throughout the year. In general, atmospheric conditions in the mine are almost ideal for efficient operation.

WAGE, CONTRACT, AND BONUS SYSTEMS

With few exceptions all tunnels, drifts, crosscuts, raises, and winzes are driven by contract. Some stoping is done by contract, but to date no general agreement satisfactory to both company and miners has been reached covering all operations in the stopes.

Wages in the mine range from $5.50 to $6.50 per day. Miners in dry places and muckers in dry places are paid $6.00. This is the general wage underground and is paid timbermen helpers, motormen, skip tenders, and hoistmen. Head timbermen, miners in wet places, and winzemen are paid $6.50.

Contract rates vary from $5.00 to $8.00 per foot for drifts, $2.50 to $5.00 for raises, and $12.00 to $15.00 per foot for winzes. Factors which determine the contract price are size of face, distance the rock has to be trammed, and the amount of water which makes at the working face. No bonus has been paid for speed. Although different contractor's earnings vary greatly, when taken as a whole the rate will amount to 25 per cent above the average wage. The advantage to the company lies in the speed of performance rather than in any saving in cost, for the cost per unit of work done is about the same whether done by contract or by wage system.

Contract prices in stopes are based upon tons of ore broken. This has not proved entirely satisfactory because the contractor continuously breaks into the walls of the stope in order to get more tonnage. The cost per ton of mining is lower, but the dilution of ore so broken more than offsets the difference in cost. A scale is being worked out based upon the maximum width to be broken. The contractor will be paid for the advance along the vein at this width. Where the vein widens appreciably, the prices per foot advanced will be raised, but only when sanctioned by the superintendent.

Measurements and settlements are made once a month.

FIRE HAZARDS

There are very few fire hazards in the mine, since all working places are damp and all timber is saturated with water.

SAFETY METHODS

No formal Safety First organization has been formed, but foremen and shift bosses are required to instruct all men under them how to perform their duties in a safe manner.

The preparation of primers and explosives for blasting is done by men experienced in this work.

All dangerous places, such as magazines, electric switches, and bare electric conductors, are guarded as far as possible with railings and guards. They are conspicuously marked with signs containing the word "Danger" and the nature of the hazard.

First-aid boxes and apparatus are installed at the portal of the tunnel and also at the office.

MINING COSTS

The following table gives the cost of delivering a ton of ore to the mine surface bins of the Harmony Mines Co. This includes development costs and transportation of waste as well as all costs connected with the mining and handling of ore.

Summary of Costs

Name of Mine: Harmony Mine.
Period covered: Jan. 5, 1929, to June 5, 1929.
Tons mined: 19,000.
Mining method: Shrinkage stope.

	Underground costs per ton							
	Labor	Super-vision	Air drills and steel	Power	Timber	Explo-sives	Other sup-plies	Total
Develop-ment	$0.32	$0.19	$0.03	$0.02	$0.05	$0.09	$0.05	$0.67
Mining	.65	.30	.27	.16	.18	.27	.22	2.05
Transp.	.15			.03			.03	.21
General	.16			.01			.06	.23
Total	1.28	.39	.30	.22	.23	.36	.36	3.14

Summary of Costs, in Units, of Labor, Power, and Supplies

Name of mine: Harmony mines.

Period covered Jan. 5, 1929, to June 5, 1929.

Tons of ore mined: 19,000.

Mining method: Shrinkage stope.

A. Labor:

	Man-hours per ton
Breaking (drilling and blasting)	1.19
Mucking	.06
Haulage	.04
Supervising	.26
General	.31
Total labor underground	1.70
Average tons per man shift	4.72
Labor, percentage of total cost	53.20

B. Power and Supplies:

Explosives (lbs. per ton)	2.05
Total power (kw.-h. per ton)	10.10
Other supplies in percentage of total supplies and power	24.50
Supplies and power, percentage of total cost	46.80

NOTE: Mining and development charges were not kept separate during period covered by report.

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FEBRUARY, 1930

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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CONCENTRATOR METHODS AND COSTS AT THE HAYDEN PLANT OF THE NEVADA CONSOLIDATED COPPER CO., ARIZ.

BY

W. I. GARMS

February, 1930.

To the reader:

This is one of a new series of papers that will present the details of milling practice and itemized costs at the principal ore-dressing plants in the United States. Officials and mill superintendents of the mining companies will prepare these papers, in cooperation with the mining division of the Bureau of Mines, and in accordance with a bureau outline designed to obtain uniform presentation.

The facts thus assembled have immediate value to the Bureau's research engineers working on ore-dressing problems and will be the basis for summarized reports by the bureau on methods and costs of separate steps in milling, such as crushing, grinding, screening, conveying, and concentrating by gravity, flotation, and leaching. Publication in mimeographed form allows these papers to be issued within a short time after receipt, and it is hoped they will be welcomed as cordially by the mining public as was the series on mining methods and costs.

Here again, we are counting on the hearty assistance of the industry, and the result should be another fine example of coordinated effort between industry and government, for the good of all concerned.

Scott Turner

SCOTT TURNER,
Director.

I.C.6241.
February, 1930.

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DEPARTMENT OF COMMERCE--BUREAU OF MINES

CONCENTRATOR METHODS AND COSTS AT THE HAYDEN PLANT OF THE NEVADA CONSOLIDATED COPPER CO.[1]

By W. I. Garms[2]

INTRODUCTION

This paper, describing the milling practices at the Hayden concentrator of the Nevada Consolidated Copper Co. is the first of a series being prepared by the Bureau of Mines on milling methods and costs in various districts of the United States.

The Hayden concentrator is located at Hayden, Gila County, Ariz., 20 miles from the mine at Ray, Pinal County, Ariz., and approximately 1-1/2 miles north of the junction of the Gila and San Pedro Rivers. During the fourth quarter of 1928 the concentrator treated daily by flotation an average of 10,366 tons of ore containing 1.25 per cent of copper. The average recovery during this period amounted to 87.84 per cent of the total copper. The concentrates contained 19.38 per cent of copper.

Water for domestic and milling purposes is obtained from a system of wells in the gravel beds of the flats on the north side of the Gila River. The flats below the concentrator also afford a readily available tailings-disposal area of 1,000 acres, situated low enough so that at the present rate of plant output the tailings can be handled by gravity for 35 years. This large area allows complete clarification of the tailings water, which is returned to the river.

The mine at Ray[3] is of the underground type and employs the undercut block-caving system of ore extraction. The coarse-crushing plant is at the mine and consists of gyratory crushers and large rolls that reduce the ore to 1-in size for shipment to the concentrator.

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1 The Bureau of Mines will welcome reprinting of this article provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6241."

2 One of the consulting engineers of the U. S. Bureau of Mines, and general mill superintendent of the Hayden concentrator of the Nevada Consolidated Copper Co.

3 Thomas, R. W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz. Circ. 6167, Bureau of Mines, 1929.

Power for the mine and mill is generated at the company's steam plant, at Hayden; during the past year, however, some hydroelectric power has been purchased from the Salt River Valley Water Users' Association. The power plant is equipped for either coal or oil fuel, but the latter has been used exclusively since 1926.

ORE TREATED

The ore comes to the mill from two shafts at Ray, the No. 1 and the No. 2. The latter furnishes the larger proportion and at times the entire mill tonnage.

The ore from No. 1 mine is fairly hard; that from No. 2 mine is somewhat softer, probably as soft as average mine ores. The copper minerals are disseminated through a quartz-sericite schist gangue. The quartz is a microscopic primary constituent of the rock. Secondary quartz occurs in the schist and as gangue in stringers of chalcocite. Sericite occurs throughout the schist. Pyrite is the most abundant sulphide mineral and is thoroughly disseminated through the ore, particularly along the lines of fracturing and the planes of schistosity. Analyses of samples of schist ore from the No. 2 and No. 1 shaft workings are given by Ransome[4] as follows:

	No. 2 mine	No. 1 mine
SiO_2	69.35	68.52
Al_2O_3	12.96	15.35
K_2O	5.02	5.77
FeS_2	4.98	2.03
Cu_2S	2.85	3.18
H_2O	2.38	2.27

The chief copper mineral in the Ray ores is chalcocite, which has replaced pyrite. There is much more pyrite in the No. 2 ore than in the No. 1. There are also present small amounts of the sulphate, the carbonate, and the silicate of copper, the last ranging from copper-stained quartz to chrysocolla. The partial replacement of the pyrite by chalcocite has resulted in the presence of a considerable number of particles of what appear to be pure chalcocite but which in reality contain a kernel of pyrite within a shell of chalcocite. This condition appears to exist in all the sizes, ranging from the smallest discernible under the microscope to the 10-mesh size of material produced by the rolls. As it arrives at the concentrator the ore contains an average of 3.5 to 4.0 per per cent of moisture. A very small amount of gold and silver occurs with the copper in the ore and when concentrated amounts to approximately 0.006 ounce of gold and 0.40 ounce of silver per ton of concentrate.

BRIEF HISTORY OF CONCENTRATOR OPERATIONS

The present flow sheet and scheme of operation are products of gradual evolution from the early days of gravity concentration and hydraulic classifica-

4 Ransome, F. L., The Copper Deposits of Ray and Miami, Ariz.: U. S. Geol. Survey Prof. Paper 115, 1919, p. 159.

tion. Before 1915 roll-crushing followed by Chilean mill grinding, in conjunction with the use of hydraulic classifiers, Wilfley tables, and Isbell vanners, gave an average of 70 per cent mill extraction.

The introduction of flotation in 1915 for treatment of gravity-mill tailings stepped up the extraction to 75 per cent. The addition of ball-mill grinding in 1918 improved extraction further.

From 1918 to 1924 the following improvements were made: (1) The number of gravity-concentrating machines was gradually reduced by the substitution of Deister tables for Wilfley tables and vanners; (2) hydraulic classifiers were eliminated, with the cone tanks and thickeners their use had necessitated; (3) Chilean mills were eliminated, and additional ball mills were substituted for fine grinding; (4) large-capacity, straight-drag classifiers and bowl classifiers were installed, and finer grinding and higher extraction were accomplished in consequence; and (5) xanthate-lime reagents were used in flotation in place of the oil-trona reagents.

In 1924 the local research laboratory developed a new flotation reagent known as raconite, which was the reaction product of a higher alcohol with variable amounts of carbon bisulphide and caustic alkali. It has been in use in the plant since 1926, with improved results.

In 1925 the construction of new flotation units, with longer, narrower, shallower cells than those formerly used, further increased flotation efficiency. During this year the use of tables for the treatment of fine sands was discontinued after tests indicating that there was no advantage in their operation when the total ore was crushed to a fineness represented by 10 per cent on a 100-mesh screen.

In 1926, 2-inch balls were substituted for 3-inch balls in the fine-grinding mills after tests had shown their advantage.

In 1927 the operation of primary Deister tables, which were used for the treatment of coarse sands, was discontinued, with resulting economy if not any appreciable metallurgical advantage.

Matless cells of the multiple-pipe type have been tested very thoroughly at this plant but have in all instances proved slightly inferior metallurgically to the straight mat-type air cells in the treatment of Ray ores.

There has been a marked reduction in milling costs in the past 2 or 3 years, although there has been no important remodeling program. Probably the two most important factors in this reduction have been: (1) The use of coarser screens in the roll circuit has given a better balanced load to the fine-crushing and the fine-grinding departments. Seven-mesh screens, 0.047-inch wire, were substituted for the previously used 8-mesh screens, 0.063-inch wire, resulting in a marked improvement in steel and power consumption and general maintenance costs. (2) The added effectiveness of the higher alcoholic xanthates has permitted marked reduction in the number of flotation machines needed, with a resulting reduction in power and maintenance costs. Incidentally, the large classifier capacity, together with the use of lime in the ball mills and the condi-

tioning of the flotation reagent before the feed is introduced to the flotation plant, have played their part in bringing about the selection of copper minerals and the dropping of iron minerals, thus eliminating a certain amount of flotation capacity which had been employed in the needless recovery of pyrite. Costs have also been reduced in the dewatering plant, again due indirectly to the flotation reagent making flocculated easily dewatered mineral concentrates.

The following tabulation gives a few outstanding items, showing where costs have been reduced:

	Fine crushing			Flotation	
Year	Roll steel, pound per ton	Kw.h. per ton	Total department costs per ton	Kw.h. per ton	Total department costs per ton
1926	0.246	4.115	$0.13207	2.753	$0.12378
1927	.192	3.684	.10481	2.588	.11446
1928	.143	3.237	.08128	1.763	.09406

Extractions and grades of concentrates during recent years are not comparable because of the varying percentage of ores received from the No. 1 and No. 2 shafts. The No. 2 ore gives bulk-flotation concentrates averaging 12 per cent of copper and selective flotation concentrates, under the present methods of treatment, averaging 19 per cent of copper, whereas the No. 1 ore gives bulk-flotation concentrates containing 20 per cent of copper and selective flotation concentrates containing 30 to 35 per cent of copper.

Present method of concentrating

The concentrator is divided into eight sections, each with a capacity of 1,500 tons per day. The sections are flexible enough to permit the treatment of as much as 2,000 tons each, although with poorer metallurgical recovery due to resulting coarser grinding.

Briefly, the operations at the coarse-crushing plant and at the concentrator are as follows:

(1) Coarse crushing by gyratory breakers and large rolls to 1-inch size for shipment to the concentrator.

(2) Fine crushing in a roll circuit from through 1-inch to through 10-mesh size.

(3) Fine grinding in a ball-mill circuit to 10 per cent on a 100-mesh screen.

(4) Concentration by flotation of fine-grinding product, producing finished concentrates and reject tailings.

(5) Dewatering of flotation concentrates in settling bins and filters.

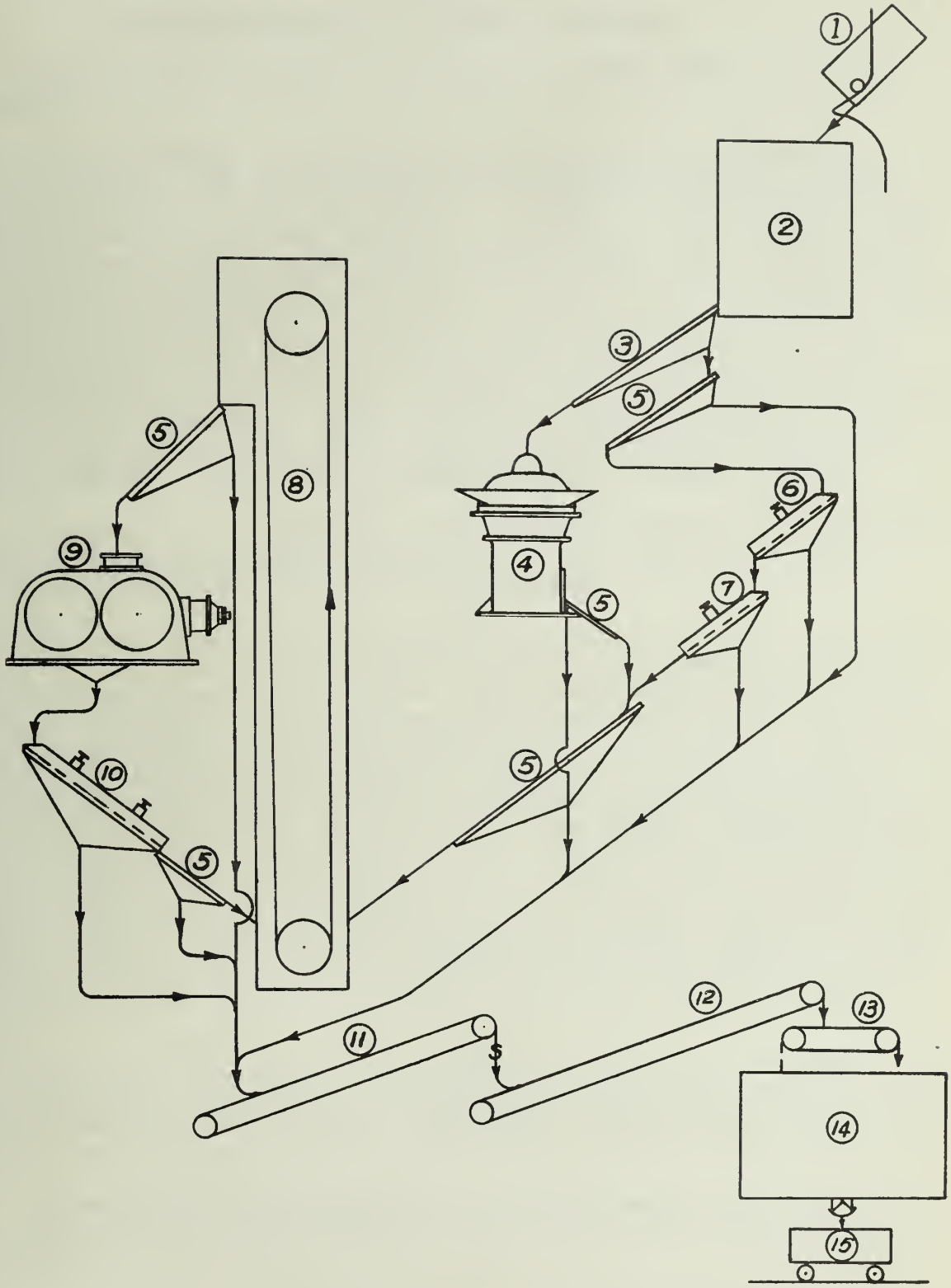


Figure 1.– Coarse crushing plant

Coarse breaking and intermediate crushing

Figure 1 is a flow sheet of the coarse-crushing plant. The legend is as follows:

1, Two ore skips, 12½ tons each; 2, receiving bin, 400 tons; 3, stationary screens, 3-inch square opening, 1¼-inch rods; 4, two No. 8 Gates gyratory crushers (348 r.p.m.); 5, stationary screens, 1¼-inch square openings, ½-inch rods; 6, two 4-foot hummer screens, 1-inch square openings, ¼-inch rods; 7, one 4-foot hummer screen, 1-inch square opening, ¼-inch rods; 8, two elevators, 36-inch belt (speed of belt, 432 f.p.m.), 18 by 9 by 9 inch buckets spaced 12 inches center to center, 49 feet center to center; 9, two 72 by 20 inch rolls (93 r.p.m.); 10, two 7-foot hummer screens, 1-inch square opening, 5/16 -inch rods; 11, incline conveyor, 42-inch belt (speed of belt, 320 f.p.m.), 78 feet center to center; 12, incline conveyor, 42-inch belt (speed of belt, 320 f.p.m.), 195 feet center to center; 13, shuttle conveyor, 42-inch belt (speed of belt, 332 f.p.m.), 103 feet center to center; 14, ore bin, 12,000 tons; 15, 30-ton ore cars; S, sampler, cut interval 3 minutes 4 seconds, opening 4 inches.

The mine ore is broken and coarse-crushed at Ray for shipment to the concentrator. There is a coarse-crushing plant at the head of each shaft in which the breaking and intermediate crushing is done.

The size of material delivered to the gyratory breakers is governed by large grizzlies at the tipples on each haulage level of the mine; the grizzly openings are 16 by 18 inches in the clear. Each coarse-crushing plant contains two No. 8 Allis-Chalmers gyratory breakers and two sets of 72 by 20 inch rolls to handle the shaft output, which has exceeded 13,000 tons per day at the No. 2 shaft. The flow of the ore is as follows: Run-of-mine ore passes over grizzlies with openings of 3 inches, the oversize of the grizzlies going to gyratory breakers. The undersize of the grizzlies passes over Hummer vibrating screens with 1-inch square holes. The undersize of the vibrating screens goes to storage bins for shipment to the concentrator. The oversize of the vibrating screens joins the crushed product of the gyratory breakers and after passing over stationary screens with 1-1/4-inch square openings goes to elevators andthence to the rolls set at 3/8 inch. The rolls are in closed circuit with vibrating screens following the rolls and stationary screens in the launders from the elevators to the rolls.

The gyratory breakers are belt-driven and yield a 3-inch product. The rolls are belt-driven and operate at 95 revolutions per minute. The steel consumption is 0.01 pound per ton of ore. The Hummer vibrating screens in closed circuit with the rolls are 4 by 7 feet; the other screens are 4 by 5 feet.

Screen sizes of the product of the coarse-crushing plant are shown in Table 1, and distribution of the copper in this product, by screen sizes, is shown in Table 2.

TABLE 1. - Screen sizes of concentrator head and products

Size of screen opening or mesh	Head to concentrator, coarse-crushing plant product, 1,500 tons per section per day, 3.5 to 4.0 per cent moisture		Product from 42-inch rolls of fine-crushing department, 1,500 tons per section per day, 55 per cent solids		Product from fine-grinding department, head to flotation cells, 1,500 tons per section per day, 32 per cent solids		Flotation tailings		Flotation concentrates	
	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent
+0.742-inch	5.64	5.64								
.525-inch	16.64	22.28								
.371-inch	14.89	37.17								
3-mesh	12.21	49.38								
4-mesh	6.96	56.34								
6-mesh	6.14	62.48								
8-mesh	5.08	67.56								
10-mesh	4.26	71.82	1.1	1.1						
14-mesh	2.95	74.77	5.2	6.3						
20-mesh	2.21	76.98	6.1	12.4						
28-mesh	2.53	79.51	8.9	21.3						
35-mesh	2.16	81.67	8.8	30.1						
48-mesh	1.53	83.20	6.2	36.3						
65-mesh	1.35	84.55	6.2	42.5	2.2	2.2	2.3	2.3	0.5	0.5
100-mesh	1.52	86.07	5.5	48.0	8.0	10.2	8.2	10.5	4.0	4.5
200-mesh	2.03	88.10	8.6	56.6	17.3	27.5	16.6	27.1	30.1	34.6
-200-mesh	11.90	100.00	43.4	100.0	72.5	100.0	72.9	100.0	65.4	100.0

TABLE 1. - Screen sizes of concentrator head and products (Continued)

Size of screen opening or mesh	Ball-mill discharge two 7 by 10 foot ball mills per section 3,000 tons per day, 80 per cent solids		Primary classifier-bowl overflow, 675 tons per section per day, 30 per cent solids		Circulating classifier overflow 2 classifiers per section, 825 tons per section per day, 34 per cent solids		Primary classifier drag sands, feed to ball mills, 825 tons per section per day, 80 per cent solids		Circulating classifier drag sands, feed to ball mills, 2175 tons per section per day, 80 per cent solids	
	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent
0.742-inch										
.525-inch										
.371-inch										
3-mesh										
4-mesh										
6-mesh										
8-mesh										
10-mesh							2.0	2.0	0.7	0.7
14-mesh	1.3	1.3					9.5	11.5	1.1	1.8
20-mesh	1.6	2.9					12.1	23.6	2.2	4.0
28-mesh	4.2	7.1					15.1	38.7	5.8	9.8
35-mesh	7.6	14.7					16.1	54.8	10.5	20.3
48-mesh	10.2	24.9					11.2	66.0	11.8	32.1
65-mesh	13.1	38.0	1.0	1.0	3.2	3.2	10.5	76.5	19.1	51.2
100-mesh	19.1	57.1	4.0	5.0	11.3	14.5	6.7	83.2	21.6	72.8
200-mesh	18.4	75.5	10.2	15.2	23.0	37.5	7.3	90.5	17.8	90.6
-200-mesh	24.5	100.0	84.8	100.0	62.5	100.0	9.5	100.0	9.4	100.0

Table 2. - Screen analyses of product of coarse-crushing plant and product of fine-crushing department showing distribution of copper.

Product of coarse-crushing plant, concentrator heads					
Screen opening or mesh	Product per cent	Product cumulative per cent	Copper analysis, per cent	Per cent of total copper	Cumulative per cent of total copper.
Heads	100.00	- -	1.25	100.00	- -
+0.742-inch	5.64	5.64	.87	3.94	3.94
0.525-inch	16.64	22.28	.96	12.83	16.77
0.371-inch	14.89	37.17	1.01	12.08	28.85
3-mesh	12.21	49.38	1.09	10.69	39.54
4-mesh	6.96	56.34	1.15	6.43	45.97
6-mesh	6.14	62.48	1.22	6.02	51.99
8-mesh	5.08	67.56	1.27	5.18	57.17
10-mesh	4.26	71.82	1.40	4.79	61.96
14-mesh	2.95	74.77	1.47	3.48	65.44
20-mesh	2.21	76.98	1.55	2.75	68.19
28-mesh	2.53	79.51	1.70	3.45	71.64
35-mesh	2.16	81.67	1.90	3.30	74.94
48-mesh	1.53	83.20	2.13	2.62	77.56
65-mesh	1.35	84.55	2.25	2.44	80.00
100-mesh	1.52	86.07	2.50	3.05	83.05
200-mesh	2.03	88.10	2.72	4.43	87.48
-200-mesh	11.90	100.00	1.31	12.52	100.00

Product of fine-crushing department - heads to fine-grinding department					
Screen size, mesh	Product per cent	Product cumulative per cent	Copper analysis, per cent	Per cent of total copper	Cumulative per cent of total copper
Heads	100.0	- -	1.25	100.0	- -
+10	1.1	1.1	.85	.7	.7
14	5.2	6.3	.90	3.7	4.4
20	6.1	12.4	.94	4.6	9.0
28	8.9	21.3	1.05	7.5	16.5
35	8.8	30.1	1.24	8.7	25.2
48	6.2	36.3	1.44	7.1	32.3
65	6.2	42.5	1.55	7.7	40.0
100	5.5	48.0	1.80	7.9	47.9
200	8.6	56.6	2.10	14.4	62.3
-200	43.4	100.0	1.09	37.7	100.0

Fine crushing and grinding

Fine crushing - roll circuit

Figure 2 is a flow sheet of one section of the fine-crushing and fine-grinding department. The legend is as follows:

1, 60-ton ore cars; 2, ore bins, 3,200 tons; 3, two apron feeders 30 inches wide, traveling 12 feet per minute; 4, conveyor, with 20-inch belt (speed, 230 feet per minute); 5, elevator, with 30-inch belt and 15 by 9 by 9 inch buckets spaced 19 inches (speed of belt, 400 feet per minute), 60 feet 6 inches center to center; 6, eight 3 by 4 foot impact screens (576 vibrations per minute), 7-mesh 0.047 inch wire; 7, two sets of 42 by 16 inch Garfield rolls (125 r.p.m.); 8, one 18-foot-diameter bowl classifier (2 r.p.m.), 8 foot by 28 foot 4 inch drag (17 strokes per minute); 9, two 8 foot by 28 foot 4-inch drag classifiers (18-1/2 strokes per minute); 10, two 7 by 10 foot ball mills (16.4 r.p.m. and 16-2/3 r.p.m.), 2 inch balls; 11, two elevators, 16 inch belt, 24 feet 4-1/2 inches center to center (369 f.p.m.), 15 by 9 by 9 inch buckets, spaced 18 inches.

There are seven of these sections, each with an average capacity of 1,500 tons per 24 hours or a total of 10,500 tons for the seven sections. The tonnage of each section may be increased to 2,000 tons by sacrificing the fineness of grinding. An eighth section, which has not been in use for several years, has recently been equipped as a test section, with a 9 by 12 foot ball mill and an 8 by 14 foot drag classifier as a substitute for the standard roll circuit, while the fine-grinding circuit is the same as the other seven sections. This section has not run long enough for a decision to be made as to its value compared with that of standard sections. It is the intention to test both rods and balls in the 9 by 12 foot mill.

The ore is removed from the bottom of the bins by apron feeders which dump on short conveyors and in turn discharge into the roll circuits of the individual sections.

Each roll circuit consists of two sets of 42 by 16 inch Garfield rolls, operating at a speed of 125 revolutions per minute in closed circuit with eight 3 by 4 foot Chino-type impact screens. Each screen has an available screening area of 8.78 square feet, which is covered with 7-mesh, 0.047-inch wire, double-crimped screen cloth. An elevator belt 30 inches wide, operating at a speed of 400 feet per minute and covered with a double row of 9 by 9 by 15 inch buckets, staggered, and approximately 19 inches apart along the length of the belt, provides for closing of the circuit. The original feed is dumped into the elevator, which discharges on the screens. The oversize from the screens passes through the rolls and into the elevators. The screen undersize flows to the fine-grinding department. The crushing in the roll circuit is wet, almost 1 ton of water per ton of original ore being added at the roll chutes underneath the rolls. It is estimated that approximately 3 to 4 tons of ore is circulated for each original ton from the conveyor. Milk of lime is added to the roll circuit at the rate of approximately 2 pounds of 90 per cent burnt lime per ton of ore. This lime neutralizes the acid condition of the ore,

takes up the bicarbonate salts in the water, and otherwise conditions the pulp for subsequent adding of the flotation reagent. All the equipment is served very handily by a 20-ton capacity crane, which runs the entire length of the building.

Elevator belts ordinarily last 500 days and cause very little lost time -- 0.25 per cent of the total possible running time. Roll shells of high-carbon steel lasted an average of 53 days in 1928; a set can be changed in less than 2 hours. When a set is being changed the feed on the section is reduced approximately 40 per cent. The total lost feeder time in the fine-crushing department in 1928, due to operating delays, amounted to only 0.63 per cent of the total possible running tim. The impact screen cloth had an average life of 7.39 days. Table 1 (p. 6) shows screen analyses of the head and the finished product of the fine-crushing department. Table 2 shows the distribution of copper, by screen sizes, in the heads and the finished product of this department.

Fine grinding - ball-mill circuit

Figure 2 is a flow sheet of the fine-grinding department; Table 1 gives screen sizes of the various products received and delivered by the ball-mill circuit, and the screen sizes of the various intermediate products of the fine-grinding circuit.

The fine-grinding circuit consists of one 8 by 28 foot bowl-type drag classifier, with an 18-foot bowl, two 8 by 28 foot drag classifiers, two 7 by 10 foot cylindrical ball mills, and two short, single-bucket elevators.

The impact-screen undersize from the fine-crushing department discharges into the bowl classifier, which overflows a product of minus 100-mesh, containing 30 per cent solids. The rake discharge of the bowl classifier is split evenly between the two mills, each mill being in closed circuit, via a short elevator, with one of the straight-drag classifiers. The combined overflow product of the bowl-type and straight-drag classifiers, averaging 10 per cent on 100-mesh and 32 per cent solids, passes to the flotation plant.

The elevator belts ordinarily last 600 to 700 days. The ball-mill liners, especially those consisting of scrap rail or roll shells, last almost 2 years. The consumption of balls amounts to about 1 pound per ton of original ore. The 2-inch balls are manufactured at Tucson or Phoenix from scrap steel, preferably rails or automobile scrap plus enough scrap cast iron to make a good melt. Specifications are not given for the balls other than that they should be smooth, round, and with minimum-sized casting necks and that consumption should come within past practice of 1 pound per ton of original ore. The consumption of liners amounts to about 0.07 pound per ton. The mill load consists of approximately 16 tons of balls. The classifiers are the standard Dorr model D type and under the conditions employed at Hayden (17 to 18 strokes per minute) cause few delays and very low maintenance expense.

The original eight mills installed have rough-cast gears and are driven by 125-horse power induction motors through double-reduction gears. The second eight mills use helical gears in single reduction, with 150-horse power

synchronous motors and magnetic clutches. The speed of the mills is 16-1/2 revolutions per minute.

Flotation

Figure 3 shows the flow sheet of one standard 2,000-ton flotation machine. The legend is as follows:

12, Distributor for four fine-grinding sections; 13, 6 inch Wilfley pump; 14, flotation primary roughers - 2 cells, each 15 feet long by 30 inches wide by 24 inches deep at shallow end, bottom slope 1/2 inch to 12 inches, 26 square feet blanket area; 15, flotation secondary roughers - 6 cells, same dimensions as primary roughers, 78 square feet blanket area; 16, flotation cleaner - 1 cell, same dimensions as primary roughers, 13 square feet blanket area.

There are 28 such machines at this plant, but not all of them are equipped with enough pumps to be immediately available under the present plan of operation. At present there is enough equipment to take care of a maximum of 20,000 tons of ore per day with plenty of spare pumps and blowers.

Baker 4-ply air blanket cloth is used on the flotation cells, and in 1928 the average life of a blanket was 51.9 days. There are six 10,000-cubic foot and one 3,000-cubic foot Root blowers available for service. A large blower can furnish air for three and one-half flotation machines having a capacity of 7,000 tons of ore. The blowers furnish air for flotation under 4-3/4 pounds of pressure. Four 8-inch and seven 6-inch Wilfley pumps are available in the flotation department.

The combined classifier-overflow product of the individual fine-grinding circuits is assembled at two points, where it enters two 26-inch pipe lines for conveyance to the flotation department.

The flotation plant is separated into two independent units, each taking the feed from four sections of the fine-grinding department. The feed may then be distributed in any desired quantities to the individual machines. It has been determined that each machine can economically handle 2,000 tons of ore in 24 hours, but owing to the size of the blower units it is frequently found convenient to vary this tonnage distribution. The feed upon leaving the distributor passes through a sampler and thence is conveyed by a launder to a pump sump; from the sump it is pumped to the head of two primary rougher cells in series. The concentrate from these two cells goes by gravity to a single cleaner cell, where a final concentrate is produced. The tailings from the primary rougher cells are piped to the head of six secondary rougher cells in series. These cells produce a waste tailing and a rougher concentrate, which, combined with the tailing from the cleaner cell, is pumped to the head of the primary roughers. This material can be pumped with the original flotation feed, and unless a special test is being conducted such is the regular practice.

Lime, raconite, and pine oil are the reagents used in the flotation operation.

The lime is fed to the plant in the form of milk of lime through two distributing systems, A and B. The A system distributes the lime to the various sections of the fine-crushing department. The lime is pumped from a large supply tank through a 4-inch header pipe, which is tapped at the primary elevator on each section. Alkalinity determinations are made on the screen undersize product, and the rate of addition of lime varied so as to maintain only a slight degree of alkalinity. The B system distributes lime to all the ball mills. System B receives its lime through a 2-inch pipe line from the 4-inch header pipe of system A. A controlled amount of milk of lime is fed through the 2-inch pipe into a 100-gallon receiving drum, from which it is pumped through a 2-inch header pipe tapped at each ball mill, and the milk of lime is fed through a bushing so that each mill receives approximately the same amount. Water is added to the milk of lime in the 100-gallon receiving drum so as to permit enough velocity through the header pipe to prevent settling and choking. The excess volume is returned to the receiving drum, where a float valve is used to regulate the inflow of new water. Alkalinity determinations are made on the flotation head and the degree of flotation pulp alkalinity controlled by the rate at which the initial milk of lime is fed at the receiving drum. The following table shows the typical alkalinity maintained at various points in the mill circuit. It is to be noted that the highest degree of alkalinity is maintained at the ball mills where the lime is most effective in depressing the floatability of the pyrite.

	Alkalinity c.c. N/28 HCO per 20 c.c. of pulp water	
	Phenolphthalein indicator	Methyl orange indicator
Screen undersize	0.3	1.1
Bowl-classifier overflow	Trace	.5
Ball-mill discharge	15.0	15.9
Secondary classifier overflow	10.4	10.9
Flotation heads	4.0	4.4
Flotation tailings	3.5	3.7

The raconite, a mixture of approximately 70 per cent sodium butyl xanthate and 30 per cent thiocarbonate, is pumped daily from the reagent-manufacturing plant in the form of an aqueous solution containing 1-1/2 pounds of the mixture per gallon to two mill-supply tanks, from which the solution is fed in alternation by gravity to the plant distributing systems. The distributing systems consist of (a) receiving drums into which a measured amount of reagent from the supply tanks is continuously fed and diluted with a large volume of water; (b) main distributing pipe lines connected to the receiving drums to carry the solution to the various mill sections; and (c) small pipes which lead from the main pipe line at the various sections and deliver the solution to the overflow of all bowl and drag classifiers. Two such systems are employed - one for the four north-side mill sections and one for the four south-side sections.

The receiving drum consists of a 100-gallon steel drum set on end with the top end removed. The drum is located high enough to permit gravity

flow throughout the system. A large volume of water (approximately 30 parts of water to 1 part of raconite solution) is permitted to flow into the receiving drum through a float valve, which regulates the amount. Water is thus admitted for dilution in quantities as large as can be handled by the various outlets in the distributing system. The outlets are restricted enough to permit approximately 75 per cent of the raconite to be added to the bowl overflows and 25 per cent to the circulating classifier overflows. The amount of raconite added is controlled by intermittently measuring the amount of solution fed to the distributing system from the supply tank. A pet cock is used to regulate the flow of the solution and is set to feed a certain volume during a stipulated period of time. A large proportion of the reagent is added to the primary slime, that is, to the bowl-classifier overflows, because it was discovered that by treating the reagent with the slime pulp before introducing it into the secondary classifier overflow pulp, where the major portion of the pyrite is concentrated, the reagent may be conditioned so that it has a marked selective action upon the copper minerals.

The pine oil (Yarmor steam distilled) is added to the combined classifier pulps at the primary distributors where the pulp is distributed to various pumps for delivery to the flotation machines. The pine oil is fed from an oil feeder of the revolving pulley type and the quantity regulated to suit various ore conditions. After the pine oil is added the ore pulp is subjected to considerable agitation by cascading and pumping, so that the oil is well mixed or emulsified by the time it reaches the flotation machines.

It may be considered that the primary slime is classified out for separate treatment, since the bowl classifiers overflow the major portion of this material. The alkalinity maintained in the pulp containing the primary slime is less than that maintained in the circulating classifier overflows, and furthermore, since 75 per cent of the total raconite is added to the pulp containing the primary slime the refractory minerals contained in this product have access to the major portion of this reagent before the clean sulphide minerals contained in the circulating classifier overflows come in contact with the collecting reagent by the mixing of classifier overflow products. Of the 75 per cent of raconite added as noted above to the bowl-classifier overflows that which is not taken up by the minerals in the bowl overflow is considered as being absorbed or detained by the gangue in the slime. The collecting agent so detained or modified acts selectively toward chalcocite in preference to pyrite when the gangue in which it is absorbed or detained is introduced into the pulp containing the cleaner minerals.

Table 1 (p. 6) gives screen sizes of flotation heads, concentrates, and tailings.

Table 3 presents average analyses of flotation heads, concentrates, and tailings for 1928.

Table 4 shows the distribution of copper, by screen sizes, in flotation heads, concentrates, and tailings.

Table 5 is a typical quarterly flotation-plant report covering the fourth quarter of 1928.

Table 6 summarizes general metallurgical data for 1928.

TABLE 3. - Average analyses of flotation heads, concentrates, and tailings for 1928.

	Weight, per cent	Copper, per cent			Iron, per cent	Insoluble, per cent	Silver, ounce per ton	Gold ounce per ton
		Total	Sulphide	Oxide				
Flotation heads	100.00	1.250	1.110	.140	3.35			
Flotation concentrates	5.37	20.246	19.248	.998	28.48	10.14	0.40	0.006
Flotation tailings	94.63	.171	.079	.092	1.92			

TABLE 4. - Distribution of copper, by screen sizes, in average flotation heads, concentrates, and tailings for 1928

Product	Screen size, mesh	Per cent	Cumulative per cent	Copper analysis, per cent		Per cent of total copper	Cumulative per cent of total copper
				Total	Oxide		
Flotation heads	Total product			1.25	0.14		
	+65	2.2	2.2	.37	.07	0.6	0.6
	100	8.0	10.2	.37	.06	2.4	3.0
	200	17.3	27.5	1.14	.09	15.8	18.8
	-200	72.5	100.0	1.40	.20	81.2	100.0
Flotation tailings	Total product			.17	.09		
	+65	2.3	2.3	.26	.04	3.5	3.5
	100	8.2	10.5	.17	.04	8.1	11.6
	200	16.6	27.1	.15	.04	14.5	26.1
	-200	72.9	100.0	.17	.12	73.9	100.0
Flotation concentrates	Total product			20.25	- -		
	+65	.5	.5	8.12	- -	.2	.2
	100	4.0	4.5	8.07	- -	1.6	1.8
	200	30.1	34.6	10.87	- -	16.2	18.0
	-200	65.4	100.0	25.40	- -	82.0	100.0

TABLE 5. - Typical quarterly flotation plant report covering the fourth quarter of 1928

Product	Weight of product, tons	Analysis			Weight, pounds		
		Copper per cent	Iron, per cent	Insoluble per cent	Copper	Iron	Insoluble
Heads	953,700	1.246	- - -	- - -	23,761,440	- - -	- - -
Tailings	901,218	0.160	- - -	- - -	2,889,975	- - -	- - -
Concentrates	52,482	19.884	28.92	11.76	*20,871,465	30,356,872	12,343,862

*By difference

Operating time	days	91
Indicated sulphide copper extraction	per cent	93.46
Indicated oxide copper extraction	do.	38.72
Indicated average extraction	do.	87.84
Yarmor pine oil used	pounds	115,588
Scotch blast furnace oil used	do.	24,644
Total oil used	do.	140,232
Sodium butyl raconite used	do.	72,150
Total flotation reagents used	do.	212,382
Oils used per ton of material treated	do.	0.147
Sodium butyl raconite used per ton of material treated	do.	0.076
Total reagents used per ton of material treated	do.	0.223
Concentration ratio		18.17
Solids in feed	per cent	33.06
Analysis of oxide copper in head	do.	0.128
Analysis of oxide copper in tailing	do.	0.083
Copper saved per ton of material treated	pounds	21.88
Average weight of ore treated per calendar day	tons	10.366
Lime per dry ton of flotation head	pounds	0.677

Power consumption

	Total kw. h.	Kw. h. per ton
Flotation pumps	430,967	0.452
Blowers	1,050,569	1.101
Total	1,481,536	1,553

TABLE 6. - Metallurgical data for 1928

Total ore treated	tons	3,234,600
Days operated		362
Hours operated per day		24
Average ore treated per 24 hours per calendar year	tons	8,838
Average number of sections operating		6,427
Average ore per 24 hours per section	tons	1,375
Concentrate	do.	173,833
Average concentrates per 24 hours	do.	480
Recovery of total copper	per cent	87.06
Recovery of sulphide copper	do.	93.23
Recovery of oxide copper	do.	38.23
Ratio of concentration		18.61
Net water consumption per ton of ore	gallons	3.03
Ball consumption per ton of ore	pounds	0.99
Liner consumption per ton of ore	do.	0.07

DEWATERING AND HANDLING OF CONCENTRATES

Figure 4 gives a flow sheet of the dewatering plant. The legend is as follows:

17, 6-inch Wilfley pump; 18, 8-foot cone tank; 19, coarse concentrate bins - 270 tons capacity each; 20, Gentry crane - 4-ton Morgan; 21, 80-ton-capacity concentrate car; 22, two elevators - one used, 18-inch belt (speed of belt, 400 f.p.m.), 15 by 9 by 9 inch buckets, spaced 20 inches 63 feet center to center; 23, one 40-foot by 10 foot 6 inch Dorr settling tank; 24, one 75 by 12 foot Dorr settling tank; 25, two elevators - 1 used, 14-inch belt (speed of belt, 375 f.p.m.), 10 by 6 by 6 inch buckets, spaced 16 inches center to center, 37 feet center to center; 26, three 12 by 12 foot Oliver filters and one 14 by 14 foot Portland filter - only one used.

All the flotation-plant concentrates are conveyed by launder to a large sump, whence they are pumped by a 6-inch Wilfley pump to an 8-foot cone tank; the discharge from the cone tank goes to concrete settling bins and the overflow to a bucket elevator. The overflow from the settling bins also goes to the bucket elevator. The elevator discharge is run through a launder to a 40-foot Dorr tank, which thickens the pulp for filtering. The overflow from the 40-foot Dorr tank proceeds to a 75-foot Dorr tank, the clear overflow going to a sump, where water is returned to the mill circuit by means of pumps. The underflow of the 75-foot tank is circulated. The underflow from the 40-foot tank is elevated to the filter plant, consisting of three 12 by 12 foot Oliver and one 14 by 14 foot Portland filters, but only one is used at a time. They are covered with a 1-piece National filter cloth, which lasts about 45 days. Hard-drawn copper wire, No. 6, spaced about 1-1/4 inches, is wound around the filter cover. This wire is good for at least two windings, in other words, 90 days. The No. 8 steel formerly used was frequently subjected to pitting caused by electrolysis, which usually shortened its life to 15 or 20 days. About 22 inches of vacuum is used in filtering and about 30 pounds pressure on the blow. The filter cake, containing less than 12 per cent of

moisture, is dropped directly from the filter into 80-ton-capacity concentrate cars for switching to the smelter.

The concentrates in the concrete settling bins drain by gravity through a burlap-covered 3 by 3 foot cone set in the center of the bin bottom. The drained concentrates, containing less than 10 per cent of moisture, are loaded by a 4-ton Morgan crane into 80-ton-capacity cars for shipment to the smelter.

The cone-tank feed contains 15 to 20 per cent of solids, the overflow about 7 to 10 per cent, and the underflow about 20 to 30 per cent. The bin concentrate usually amounts to about 55 per cent of the total flotation concentrates. The filter feed varies from 55 to 70 per cent of solids. Live steam from the power plant is used to heat the pulp in the filter tank, the temperature maintained being between 118° and 125°F. This steam costs about $500 or $600 a month in summer and about $1,000 to $1,200 in winter. The average load on a filter amounts to about one-half ton in 24 hours per square foot of filtering surface. The alumina in the concentrate varies, but in 1928 it averaged about 3 per cent.

DEWATERING AND DISPOSAL OF TAILINGS

The tailings, containing approximately 2.5 tons of water per ton of solids, flow from the concentrator in a launder to the 1,000-acre disposal area. One side of the area is along the slopes of the hills; the other three sides are inclosed by dikes built of soil, gravel, and tailings material. The original dikes were built with a drag-line excavator. Subsequent raising of the dikes has been done with the excavator and a tractor-drawn scraper. The area is divided into three ponds, two of which are always ready to receive tailing, while the third is dry and the dikes thereof are being raised. The packing that results from passage of the tractor makes a good dike out of the tailings material. Because of the large tailing-disposal areas the overflow waters therefrom are perfectly clear, and there is no problem of stream pollution. Tailings water, other than that which returns by seepage to the gravel beds, is not reclaimed.

The tailings flume from the mill to the pond is 5 feet, 10 inches wide and slopes 2.4 to 2.6 per cent.

Before the advent of fine grinding and the use of the lime circuit in flotation dust storms from the tailings ponds were quite frequent. The tailings then reposed at a steeper angle, with the result that portions of the piles were dry and exposed to the winds. The tailings now being produced lie practically flat, and any dry spots that do occur are coated with a crust of claylike slime. This coat does not dust unless broken by passage of the equipment being used in raising the dikes.

There is virtually no breakage of dikes, as the water impounded behind them is never more than 1 foot in depth and ordinarily averages 6 inches.

CONVEYING, ELEVATION, AND MOVEMENT OF ORE THROUGH PLANT

As stated before, the crude ore is conveyed from underneath the ore bins to the elevator for each section. The conveyor is 25 feet long, center to center of pulleys, and 20 inches wide, and travels 260 feet per minute. The belt lasts about 3 years, as it is subjected to very easy service.

The crude ore, combined with the roll product, is elevated about 65 feet with a 30-inch bucket elevator. This circuit is wet, and it is estimated that in a standard section 1 such an elevator lifts about 10,000 tons of ore and water in 24 hours. This system is considered a very economical means of lifting pulp, as it costs only about 1.5 cents per ton of original ore to do the work. This includes about 0.67 cent for power; one man on each operating shift is charged against this service.

Launder chutes in the roll and screen circuit are lined on the bottom with adamant silica blocks and on the sides with old elevator belting.

Two launders convey the screen undersize of each section to the bowl classifier. These launders are 15 inches in width, slope 1-13/16 inches to the foot, and are lined on the bottom with cement blocks, which are made up in advance and set in when replacements are necessary. Such bottoms last at least 2 years.

Two launders convey the ball-mill discharge to the circulating classifier elevators. They are 13 inches in width, slope 1-3/4 inches to the foot, and are lined on the bottom with cement, which lasts indefinitely.

The pulp discharged from each ball mill is elevated about 25 feet with a 16-inch bucket elevator at a cost of about 0.5 cent per ton of original ore milled. This includes about 0.2 cent for power. It is considered more economical to elevate this pulp than to pump it.

The launder carrying the classifier feed from the circulating elevator to the classifier is 15 inches in width and slopes 2 inches to the foot. It is lined on the bottom with cement blocks and lasts indefinitely. Cast-iron pipes 8 to 20 inches in diameter take the classifier overflows to the 26-inch wooden pipe lines. The cast-iron lines are automatically protected from wear by the precipitation of lime on the inner surface of the pipe. Although this precipitation has caused some inconvenience in restricting the flow of the pulp the trouble has not yet been serious during the 5 years that the pipe line has been in service.

The feed to the flotation machines, combined with the cleaner tailings and the secondary rougher concentrates, is pumped through a distance of about 150 feet and to a height of approximately 45 feet by 6 and 8 inch Wilfley pumps. Under the conditions at Hayden an 8-inch pump will handle approximately 4,000 tons of solids per day in a pulp running approximately 70 per cent of moisture. The wear on pumps is quite variable, depending a great deal upon the quality of the parts and the character of ore handled. The shells last 60 to 90 days and the impellers and follower plates about 35 to 50 days. Operation of the flotation pumps costs about 0.85 cent per ton of original ore, which includes about 0.60 cent for power.

The flotation-concentrates pulp, containing not more than 20 per cent of solids, is pumped by a 6-inch Wilfley pump approximately 100 feet and to a height of approximately 30 feet into an 8-foot cone tank. This pumping is included in the above flotation-pumping operation cost. The shell lasts 110 to 140 days and the other parts 60 to 80 days.

The concentrate cone-tank discharge is conveyed by a launder to settling bins and the overflow to a bucket elevator which elevates the pulp about 65 feet, from which point it is run in another launder to a 40-foot Dorr settling tank. The bucket belt gives about 3 years of operating life. Two elevators are installed for this service, but only one is used at a time. The Dorr-tank discharge, about 60 per cent of solids, is elevated to the filters through a lift of about 36-1/2 feet. The belt gives about 3 years of operating life, there being one spare. The cost of elevating in the dewatering plant service is less than 0.1 cent per ton of original ore, and nearly all of this is power.

Sampling and accounting for copper

The ore as received from the mine is weighed on railroad scales by the Southern Pacific Railroad Co. A moisture sample is taken while the ore is being dumped into the bins. The bins are measured every morning at 7.30, and from such measurements and the receipts between measurements the dry tonnage milled for the 24-hour day is determined.

The ore is not sampled for copper content until it has been put through the fine-crushing and fine-grinding departments and is in the form of pulp ready for distribution to the flotation machines. There is no doubt that fine ore, well mixed, affords more accurate sampling than coarse ore fed from large bins onto a conveyor. The pulp as fed to the flotation machine is a very homogeneous mixture; therefore, automatic samplers which cut the various streams from the main distributors at intervals of 6 minutes obtain very reliable samples. (See location of sample cutters marked S on fig. 3.) At least four of these samples are obtained for each 8-hour shift, making 12 samples in all for a 24-hour run. Each of these samples is assayed in duplicate, and a composite head assay for each shift is thus obtained.

The dry weight of ore milled for 24 hours times the average copper assay gives the number of pounds of copper in the mill heads to be accounted for in the concentrates and tailings.

As in the case of the heads assay, very accurate determinations of copper content in the tailings and concentrates are made for the 24-hour run. Automatic cuts of the general streams of tailings and concentrates at 6-minute intervals give reliable samples.

The indicated extraction in per cent is calculated from the formula $\frac{C\,(H - T)}{H\,(C - T)}$,

where, H = assay of the heads
C = assay of the concentrates
T = assay of the tailings

The indicated ratio of concentration is calculated from the formula $\frac{C - T}{H - T}$. This ratio divided into the tons of ore milled gives the indicated tons of concentrates produced.

The indicated extraction times the copper in the heads gives the indicated recovery in pounds of copper. The difference between the pounds of copper in the heads and the indicated recovery gives the indicated pounds of copper in the tailings.

The concentrates produced are segregated into coarse and fine products, the coarse going to settling bins and the fine to the filtering plant. The segregated products are shipped in separate cars to the Hayden plant of the American Smelting & Refining Co., where they are weighed and sampled in the presence of a mill representative. Duplicate moisture and assay samples are delivered to the mill for check purposes. Concentrate pulps, for which the smelting company and the mining company have determined different assay values that can not be settled by agreement, are submitted to umpire assayers.

The pounds of copper received by the smelting company, as determined from weights and moistures and "split" or umpire assays, represent the actual recovery for the mill.

Auger samples of the concentrates as produced at the filter and bins are taken as a check on the smelter sampling. This is also a further check on the accuracy of the general assay of concentrates made on the flotation-plant product. The concentrates in the bins are measured every morning, and the weight of the filtered and bin products loaded into the cars during the previous 24 hours is estimated, affording a gauge of the daily production of concentrates to compare with the indicated tonnage, as determined by the formula given for the ratio of concentration. As a variable amount of concentrates is left in the pump sump each morning, together with a variable density of pulp in the filter thickeners, the daily check is not as close as is one taken over longer periods. Hence, every 10 days a complete inventory of the copper actually paid for by the smelter, in transit, in bins, and in thickeners is made, which parallels quite closely the indicated figure obtained by assays. As an example of this close accounting the recovery results for the last 4 years follows:

Year	Indicated copper recovered, pounds	Actual copper recovered, pounds
1925	76,908,411	76,947,957
1926	71,014,451	70,960,335
1927	65,454,928	65,579,968
1928	70,458,066	70,387,501
Total	283,835,856	283,875,761

The above indicates a difference of only 39,905 pounds between the indicated and actual copper recovered out of a total of nearly 284,000,000 pounds of copper, a difference of less than 0.02 per cent.

In addition to the sampling of the general heads, tailings and concentrates, six automatic samplers for tailings and five for concentrates have been installed on individual flotation machines for testing. The fine-crushing department is equipped with titrating apparatus for checking alkalinity in the roll circuit; the fine-grinding department is equipped with balances for determining the percentage of solids in the classifier circuit; the flotation department is equipped with both titrating apparatus and balances for alkalinity and density determinations.

Method of determining oxidized copper

The method used at the Hayden plant is a modification of that outlined in Bureau of Mines Technical Paper 198 and consists of treating a sample of ore weighing 5 grams with 100 cubic centimeters of a 2.5 to 3.0 per cent solution of sulphur dioxide in a sealed bottle. The bottle and contents are agitated for 2 hours, then the residue is filtered off and 3 to 4 c. c. of concentrated sulphuric acid is added to the filtrate. The filtrate is boiled down to a volume of about 50 c. c.; the copper is then precipitated on an aluminum coil and determined by the iodide method.

WATER SUPPLY

The source of the water supply is about 1 mile from the mill; the difference in the head between the pump station and the mill is 250 feet. A very complete article on the pumping plant was read at the meeting of the Arizona Chapter of the American Mining Congress at Ray on March 8 and 9, 1926.[5]

The water is pumped by air-lift pumps from a series of shallow wells to a common reservoir. The actual lift of water from the wells varies from a few feet to 40 feet. The reservoir is at such an elevation that a suction head on the plunger pump valves is produced. The pumping plant proper delivers the water from the first reservoir to the power plant and to domestic reservoirs, and consists of three motor-driven 16 by 18 inch vertical quintiplex plunger pumps, with a rated capacity of 3300 gallons per minute each; and two motor-driven 10 by 12 inch vertical triplex plunger pumps of a combined capacity of 1,400 gallons. The latter two are domestic water pumps.

The quintiplex pumps deliver the water to a small reservoir at the power plant, whence it is pumped to the mill reservoir by means of a Nordberg steam pump. The distance from the pump station to the power-plant reservoir is 6,394 feet and the lift is 186 feet. The distance from the power-plant reservoir to the mill reservoir is 2,500 feet and the lift is 85 feet to the spillway. The water flows by gravity from the mill reservoir to the mill, a distance of 2,300 feet; the difference in elevation from the mill-reservoir spillway to the highest point in the concentrator water system is 25 feet. The usual practice is to by-pass the water from the Nordberg pump directly to the concentrator. The mill reservoir has a rated capacity of 5,800,000 gallons.

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5 Johnson, Nels, Water Development for the Hayden Plant: Am. Min. Cong. Jour., Arizona Chapter, Ray Meeting, March 8 and 9, 1926.

The water is pure enough for domestic use but has to be treated for power-plant use due to the presence of mineral salts. At the concentrator some of these salts are precipitated out by the lime used for flotation purposes, thereby really nullifying the value of nearly 40 per cent of the lime used. Pumping the primary water used at the concentrator requires about 1.3 kw.h. per ton of ore milled. The initial expense for water in 1928 was 2.07 cents per ton of ore milled; this item includes the cost of the power.

The only water recovered at the mill is that from the plant where the concentrates are dewatered, and this water costs about 0.25 cent per ton of ore milled, including the cost of 0.15 kw.h. per ton of ore.

All water used in the town site is charged against mill operation and is included in the above cost. During 1928 such water amounted to 12,100,000 gallons per month or about 0.2 cent per ton of ore milled.

Table 7 summarizes the costs per ton of ore treated for 1928.

Table 8 shows the distribution of power for 1928.

Table 9 gives consumption of heavy supplies and ore treated per man shift for 1928.

TABLE 7. - Summary of costs per ton of ore treated for 1928

Tons of ore treated, 3,234,600. Tons of concentrate produced, 173,833.

	Operating labor	Supplies	Repair labor	Repair supplies	Power	Miscellaneous	Total
Coarse-crushing gyratory breakers and 72-inch rolls	$0.0076	$0.0087	$0.0053	$0.0019	$0.0041	$0.0022	$0.0298
Fine crushing, 42-inch rolls	.0134	.0211	.0087	.0051	.0301	.0028	.0812
Fine grinding, 7 by 10 feet ball mills	.0081	.0224	.0066	.0044	.0346	.0017	.0778
Flotation concentrating	.0150	.0465	.0034	.0019	.0163	.0109	.0940
Dewatering of concentrates	.0044	.0015	.0025	.0032	.0023	.0034	.0173
Assaying and sampling	.0059	.0034	- - -	- - -	- - -	.0004	.0097
Disposal of tailings	.0012	.0010	.0072	.0023	.0008	.0015	.0140
Water supply	.0025	.0023	.0012	.0017	.0139	.0016	.0232
Research	.0017	.0010	.0014	.0019	- - -	.0034	.0094
Miscellaneous	.0091	.0004	- - -	.0043	.0016	.0272	.0426
Totals	.0689	.1083	.0363	.0267	.1037	.0551	.3990

Table 8. - Distribution of power for 1928

	Kw. h. per dry ton milled	Per cent of total power
Coarse-crushing department.		
Total gyratory breakers and 72-inch rolls	0.442	3.9
Fine-crushing department		
Rolls	2.307	
Elevators	.685	
Other machinery	.245	
Total	3,237	29.3
Fine-grinding department		
Ball mills	3.338	
Elevators	.271	
Classifiers	.131	
Total	3.740	33.8
Flotation concentrating department		
pumps	.521	
Blowers	1.237	
Auxiliaries	.005	
Total [1]	1.763	16.0
Concentrate dewatering		
Filters	.007	
Pumps and compressors	.138	
Elevators	.067	
Auxiliaries	.031	
Total	.243	2.2
Water		
Initial	1.280	
Recovered	.220	
Total	1.500	13.6
Industrial buildings, excavator, and auxiliaries	.129	1.2
GRAND TOTAL	11.054	100.0

[1] Flotation power for first 8 months of 1929.

Pumps	0.645
Blowers	.819
Auxiliaries	.004
Total	1.468

TABLE 9. - Consumption of heavy supplies and ore treated per man shift for 1928

Department and item		Consumption per dry ton milled
Fine-crushing department		
Roll steel including waste	pounds	0.1428
Fine-grinding department		
2-inch semisteel cast balls	do.	.99
Flotation department		
Lime	do.	5.28
Yarmor pine oil	do.	.14
Scotch blast furnace oil	do.	.01
Sodium butyl raconite	do.	.072
Average life of roll shells	days	53.14
Average life of impact screens	do.	7.39
Average life roll circuit elevators	do.	542
Average life ball mill circuit elevators	do.	704
Average life ball mill liners	do.	630

Ore treated per man per shift tons

Total operation including power plant	Mill operation and repairs	Total payroll, including power plant	Total payroll, minus power plant
30.23	35.24	28.22	32.62

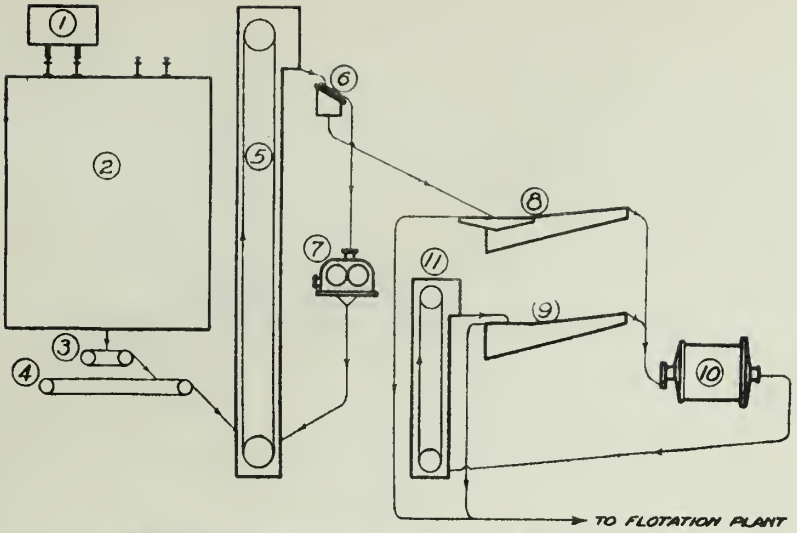


Figure 2.– Flow sheet for fine crushing and fine grinding

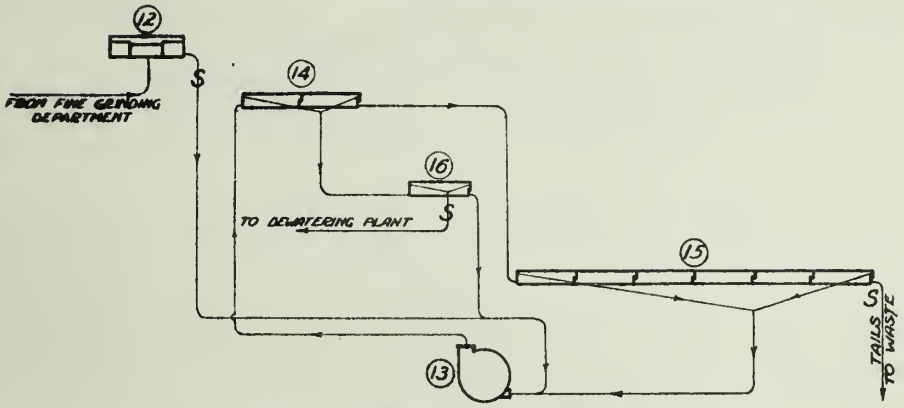


Figure 3.– Flow sheet of one unit of flotation plant

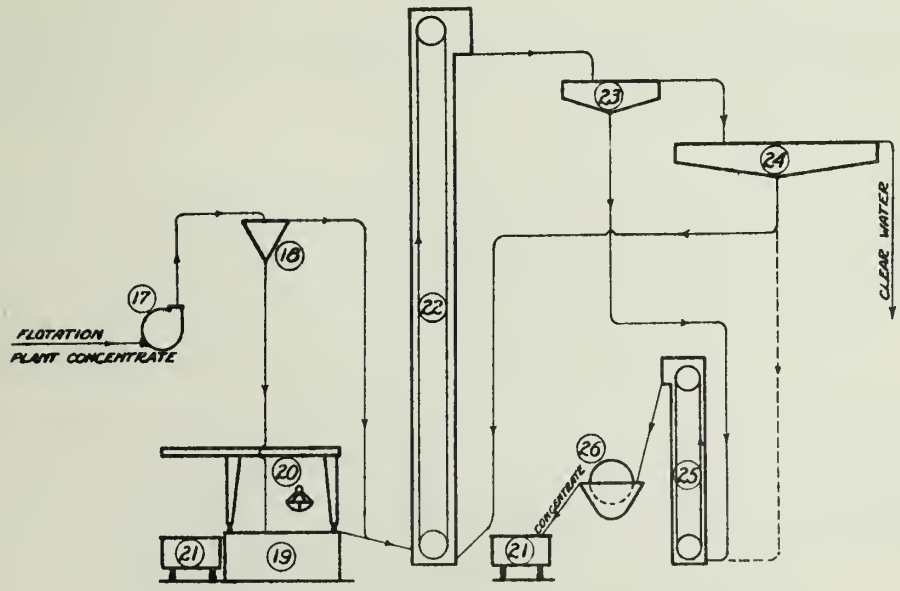


Figure 4.– Flow sheet for dewatering and handling of concentrates



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SAFETY IN UTAH COAL MINING AS AFFECTED BY HAULAGE

By D. J. Parker[2]

One writer has stated that causes of accidents may be classified as (a) mechanical or physical, (b) the human element, and (c) conditions of the workers' environment. The relative value of "b" and "c," the human element and the workers' environment, in accident causation, as compared with strictly mechanical or physical causes of accidents, is given by one writer as 75 to 25 and by another as 90 to 10. The latter ratio is probably more nearly correct with respect to accidents from underground coal mine haulage.

Forgetting for the moment the mechanical or physical aspects of the case, the method of dealing with the human element merits a good deal of emphasis and elucidation. Some of the factors entering into the human element are irrationality, ignorance, overconfidence, carelessness, indifference on the part of both employer and employee, inattention, fatigue, improper placing of individuals, chance-taking, undue hurry, improper dress, and mental or physical defectiveness of employees. Of these, carelessness is perhaps responsible for more accidents than all of the other factors combined, and is the more difficult to overcome because its consequences often lack immediate moral significance.

Any safety program, whether it deals with the entire organization or some particular phase of the operation, such as haulage, that is not founded on the three fundamental principles - education, supervision, and discipline - must of necessity fall far short of its goal.

While accidents from falls of roof and coal are responsible for about half the fatalities occurring annually in coal mines, haulage is taking a heavy toll of life, and accidents due to this cause are ever increasing. The number killed each year in our coal mines from this cause usually amounts to more than 400 men and constitutes around or over 17 per cent of the total underground fatalities from all causes. In addition, there are approximately 50 fatalities annually due to surface-haulage accidents.

The principal causes of underground-haulage accidents may be classified in their order of importances as follows: Cars or locomotives running over men, men squeezed between

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6242."

2 - District engineer, U. S. Bureau of Mines.

cars or locomotives and ribs, runaway cars, falling from trips, men caught between cars or locomotives and roof, switching and spragging, and coupling cars.

In the study of accidents from falls of roof and coal the United States Bureau of Mines has devised an index system of comparing accident rates in the various States. By using this system it is possible to compare haulage as well as other accident rates in one State against another. In the following tables the figure 100 represents the average fatality rate from haulage accidents in the United States. Any State which has a figure less than 100 shows that State to be better than the average, in so far as haulage accidents are concerned, and a figure in excess of 100 indicates that that State nas a more unfavorable record than the average. In the United States the records show that underground-haulage accidents resulted in the deaths of an average of 0.372 men per 1,000,000 hours of exposure during the eight-year period, 1921 to 1928, inclusive.

Table 1.- Index numbers comparing coal-mine fatality rates per million man hours of underground exposure, 1921-1925

HAULAGE UNDERGROUND

State	Index Numbers	State	Index Numbers
Missouri	16	Washington	105
Kansas	19	Illinois	107
Arkansas	21	Kentucky	112
Texas	34	Indiana	116
Iowa	41	North Dakota	116
Oklahoma	59	New Mexico	122
Maryland	65	Michigan	124
Virginia	76	Montana	131
Pennsylvania (bituminous)	78	Wyoming	145
Pennsylvania (anthracite)	79	Colorado	151
Tennessee	86	West Virginia	178
Alabama	92	Utah	240
Ohio	101		

Table 2.- Index numbers comparing coal-mine fatality rates per million man hours of underground exposure, 1922-1926

HAULAGE UNDERGROUND

State	Index Numbers	State	Index Numbers
Kansas	12	Alabama	100
Missouri	18	Illinois	103
Arkansas	20	Kentucky	111
Texas	36	Michigan	112
Iowa	41	New Mexico	117
Oklahoma	63	Indiana	123
Pennsylvania (anthracite)	74	Montana	124
Pennsylvania (bituminous)	77	Washington	144
North Dakota	77	Wyoming	158
Tennessee	86	Colorado	163
Virginia	90	West Virginia	173
Maryland	92	Utah	203
Ohio	95		

Table 3.- Index numbers comparing coal-mine fatality rates per million man hours of underground exposure, 1923-1927

HAULAGE UNDERGROUND

State	Index Numbers	State	Index Numbers
Kansas	6	Virginia	105
Missouri	38	Maryland	107
North Dakota	43	Kentucky	108
Iowa	45	Indiana	121
Arkansas	58	Washington	134
Pennsylvania (anthracite)	69	New Mexico	140
Pennsylvania (bituminous)	76	Montana	149
Oklahoma	80	Colorado	161
Tennessee	93	Georgia and North Carolina	163
Ohio	96	Wyoming	164
Michigan	96	West Virginia	174
Illinois	102	Utah	187
Alabama	102		

Table 4.- Index numbers comparing coal-mine fatality rates per million man hours of underground exposure, 1924-1928

HAULAGE UNDERGROUND

State	Index Numbers	State	Index Numbers
Texas	23	Alabama	101
Missouri	48	Kentucky	101
Iowa	55	Michigan	110
Pennsylvania (anthracite)	66	Maryland	113
Pennsylvania (bituminous)	80	Virginia	120
Arkansas	86	Washington	140
Oklahoma	87	Colorado	145
North Dakota	89	Wyoming	146
Tennessee	91	New Mexico	157
Illinois	95	Montana	163
Ohio	97	West Virginia	169
Indiana	100	Utah	187

The four foregoing tables give the index numbers comparing coal-mine fatality rates per million man hours of underground exposure over the periods 1921-1925, 1922-1926, 1923-1927, and 1924-1928, inclusive, with respect to underground haulage.

On this basis it will be noted that Utah has the most unfavorable accident rate of

any State in the Union. The index numbers for this State are 240, 203, 187, and 187, respectively, for the four periods covered, as against the index number of 100 for the entire country. Attention is directed to the fact, however, that the trend of fatal accidents in Utah is downward.

Another writer has stated that 2 per cent of all coal-mine accidents are not preventable, while 98 per cent can be prevented; and of this 98 per cent, 88 per cent are chargeable to inadequate supervision and 10 per cent to physical failures including physical hazards.

Due to adverse natural physical conditions existing in the coal mines of Utah, it is believed that the 10 per cent ascribed to physical failures is perhaps too low. In order properly to appreciate such conditions it must be remembered that most of the coal beds in this State are pitching, some of them have a maximum thickness of 28 feet, in some instances the overlying strata reach a maximum depth of 2,500 feet or more, and that practically every mine in the state "pulls" pillars.

The excessive headroom available in the operation of such thick beds has been conducive to adoption of increased capacity of rolling stock with attendant increased size and weight of both load and empty. Trains composed of 50 cars, varying in capacity from 8,000 to 10,000 pounds per car are handled regularly on the main haulage of one mine.

Pitching beds not only increase the hazards in connection with mechanical haulage, but they also accentuate similar hazards with respect to animal haulage. Particularly is this true where cars of large capacity are used. In the mines of Utah animals are used singly to a considerable extent for gathering purposes, and where the rooms are driven to the raise, shafts, instead of the customary tail chain, are used.

The shafts which are U-shaped are constructed in one piece of 1¼-inch round iron, and are attached to the draw bar by means of a steel bar or "gun" slightly S-shaped. The "gun" is detachable from the rigid crosspiece connecting the two shafts. The use of shafts is advantageous in that it permits the animal to exert considerable back pressure on the car, where brakes, sprags, or "sand" rails are insufficient to retard the down-grade momentum of the loaded or empty car.

Another unusual safety practice in Utah in connection with slope or incline haulage is the use of coupling pins equipped with catches so arranged that it is not possible for the pins to work out while trains are in transit. In fact, it is impossible to pull the pins out of the links and draw bars without disengaging the hook. The pin is removed by pulling upward on a ring fastened to the hook. The lower end of the catch projects underneath the end of the draw bar so as to hold the coupling pin in place.

In order to protect against accidents due to breakage of trips from failure of the coupling pins or links or from failure of the draw bars, most of the mine cars in Utah coal mines have attached on each side at the end of every car a chain which is rigidly attached to a longitudinal horizontal iron strap; the loose end of the chain can be hooked to the end

of this strap so that if the coupling pin, coupling link, or draw bar should fail, the car will remain an integral part of the trip, provided the safety chains are always hooked. These chains have prevented many wrecks and the other kind have failed to prevent many wrecks because of the failure of the attendants to hook the chains at all times.

The safe, rapid and economic transportation of coal can be brought about only by proper maintenance of track, other things being equal. The tendency at present is toward rapid transit, which requires heavier rails and greater maintenance-of-way efficiency.

In modern mines, rail weights from 60 to 80 pounds are frequently used, the individual rail sections being joined effectively by steel angle bars and supported on large ties. On main-line haulage slopes and inclines, it is decidedly advantageous to space ties not to exceed 24 inches, center to center. The size of such ties should be not less than 6 by 6 inches in cross section. All tracks, but especially those on main lines, should be well ballasted, aligned, and surfaced. All too frequently track alignments are made to conform to the horizontal or vertical irregularities or windings of the entries. Tracks well laid with suitable materials and properly maintained will not only "make for" greater efficiency, but will have a decided bearing on accident reduction. There is an apparent tendency on the part of many operators after the track is laid, to give it no further attention until the occurrence of a wreck, perhaps culminating in a serious or fatal accident, necessitates immediate and often costly repairs.

Greater safety and efficiency can be brought about by a definite system of inspection for all haulage equipment and the employment of a sufficient number of skilled workmen to keep the track in first-class condition. Proper maintenance of room tracks is of equal importance. After a track is laid it should be kept clean and maintained in accordance with carefully determined standards. The standardization of track elements, layouts, and maintenance are important factors in any accident-prevention program that has to do with coal mining. Mine shales, crushed stone, if available, cinders and adobe make excellent material for track ballast. Inflammable material such as bone or fine coal should not be used for this purpose, because its use adds definitely to the explosion hazard of the mine, thereby making it more difficult to maintain the requisite amount of inert material in the road-dust of a rock-dusted mine; in addition, bone and coal slack do not make efficient holding material when used as track ballast.

One of the haulage hazards due to falling over objects along haulage roads can be materially reduced by installing switch throws at a suitable distance from the rail and so arranging the throw that it operates parallel to the track. The rod connecting the throw to the bridle should be on a level, or below the headblock or tie supporting the throw. If necessary, the throw should be recessed into the rib.

The more or less common practice in mines, of allowing the motorman to "drift" up to switches or "latches" and then attempt to throw them by hand or foot while the locomotive is in motion, has resulted in many serious accidents; this practice is not necessary or even advisable and its use should be prohibited in all mines.

Diamond-shaped rerailers are of questionable value. The records show that at least one disastrous explosion was caused while attempting to rerail several loaded cars by the use of the diamond rerailer. In attempting to replace the cars on the track by pulling them over the rerailing device with an electric locomotive, timbers were knocked out so that a roof fall resulted which brought the trolley wire, carrying 250 volts d.c., in contact with one or more of the steel cars. The arc was sufficient to ignite the cloud of coal-dust in suspension, which resulted in a widespread explosion. Instead of relying on such devices of questionable safety each locomotive should be equipped with a set of rerailing frogs and the rerailing of cars or locomotives should be done with the greatest care.

All slopes and inclines should be protected by derail switches positive in action. To protect the face workers on slopes or planes driven to the dip, one company has standardized a derailing device consisting of an extra large post set at right angles to the strata to which is fastened with a collar near the top a 10-foot section of heavy steel rail. The free end of the rail rests near the center of the track when the derail is in an operative position.

In working places on the incline, all cars should be securely blocked either by rail clamps of the clevis type, or by placing one timber across the track with another at right angles to the first resting on top of it and extending under the end of the car. This type of block is not satisfactory, however, where the rails are wet or where the pitch is over 10 per cent.

Although it is difficult to overemphasize the necessity for good track, of equal importance to it is first-class rolling stock. Both are prerequisites to safety and efficiency, and must be recognized accordingly if the industry continues to indulge in the hope of a definite reduction in haulage accidents. Excellent rolling stock operated over poor track will probably soon be reduced to the same low standard of efficiency as the track, with consequent high cost of repairs. The converse is true, and as a result, track and rolling stock are generally found on a par with each other. Unfavorable conditions as to installation and upkeep of track or haulage equipment are not conducive to low haulage-accident records, with respect to either severity or frequency rates.

Frequent and systematic inspection and repairs of locomotives, cars, ropes, couplings, hoists, etc., plus rigid enforcement of company rules or standards, are indispensable to accident prevention or elimination.

One large company in Utah makes a thorough examination of all links and pins semi-annually. Following a close visual inspection for possible defects, the links and pins are subjected to the hammer test. The character of the "ring" will usually disclose the degree of soundness. They are then heated, examined again, and annealed in lime or gypsum dust. This treatment being given at least twice a year. The practice is particularly advantageous from a safety standpoint, especially where cars are handled on slopes or inclines.

The rapid and widespread introduction of electricity into the coal mines of this country during recent years has not only materially increased the explosion hazard, but has

also had a very definite bearing on increased accident rates with respect to transportation.

The contact hazard due to unguarded trolley wires is an everpresent one, and is taking annually a heavy toll of life. Trolley wires less than 6½ feet above the rail should be adequately guarded throughout their entire length or at least at all points where the necessity arises for men to pass under them. Several large metal-mining companies have found it extremely advantageous to guard trolley wires throughout their entire length, thereby reducing the contact hazard to a minimum. Boards of suitable length, canvas strips, and discarded fire hose are the most common means used to prevent accidental contact with trolley wires. Of the three the fire hose is perhaps the most effective, though a sufficient supply of fire hose is difficult to obtain unless new hose is used, and that would be very expensive. The hose is slit longitudinally and is securely attached to a board of suitable width and thickness. The ends of the hose are flared so that the trolley has free ingress from either end. The advantage of such a guard is that the trolley wire is practically enclosed or insulated at all times when the trolley is not passing through it.

Sectional switches are often found installed in such a way that the haulage crew is required to operate them while passing. Often this is done while the trip is moving at a fair rate of speed, and results in accidents. If such switches are not automatically controlled, the motorman should be compelled to bring his locomotive to a standstill before attempting to throw the switch. Furthermore, the sectional switch should be suitably guarded to prevent the person throwing it from receiving possible burns from resultant arcs.

Jumping off moving locomotives and running ahead to open doors or throw switches, a more or less usual practice of trip riders, is hazardous in gathering and assembling trains, but the practice is extremely hazardous when applied to heavy main-line traffic. This, together with the making of flying switches, should be strictly prohibited in any mine.

A prolific cause of haulage accidents is excessive speed of both loaded and empty trains. An efficient dispatching system will go far toward eliminating this unsafe practice. In the interest of safety it is believed that the maximum speed for loaded and empty trips should not exceed 800 feet per minute, and 440 feet per minute should be the limit for man and explosive trips.

Main haulage lines should be well illuminated and provided with shelter holes, and travel should be restricted to those employees whose duties very definitely require their presence thereon.

Fooling or "horse play" has long since been recognized as a hazard in industry, and the necessary remedial action to overcome it can best take the form of rigid discipline. The trolley wire, carrying its customary 250 volts, lends itself admirably to this reprehensible pastime, whether on the surface or underground; in one case an employee with a peg leg took a delight in occasionally demonstrating to an attentive and all too appreciative audience, the insulating qualities of his peg. On one particular day this person stood on the rail with his peg, raised his remaining foot clear of the ground, and seized the live trolley wire with his bare hand. While in this precarious position, the man's faithful dog

placed his nose in his master's free hand. Artificial respiration probably saved the man's life. Strange as it may seem, no one was ever able to explain to the entire satisfaction of the "chance taker" just why his peg failed him on this particular occasion.

Not so many years ago mining industry was prone to place the blame for accidents, especially those resulting in fatal injury, upon the victim of the accident. A close analysis of accident statistics over a very considerable period of time has disclosed the fact that in most cases the responsibility wholly or in part rested squarely upon the shoulders of operating officials. The degree to which the mine officials are willing to accept this responsibility may be taken as an index of progress in the matter of accident prevention. Due to faulty methods of mining, or laxity in matters of policy pertaining to safety, responsibility for accident occurrence can often be traced far beyond the officials immediately in charge of underground work.

This fact was strikingly illustrated recently in the fatal injury of a trip rider who was caught between an electric locomotive and the roof, following the derailment of the rear end of the locomotive on which the victim was riding. The locomotive stood 3 feet above the rail with only 12 inches clearance between the roof and the top of the locomotive at the point where the accident occurred. The victim was held responsible, as evidenced by the issuance of a subsequent warning "not to extend your body over cars or locomotives while under low roof." Obviously, the officials were responsible because of failure to provide adequate head room. Many instances of similar character might be cited.

Haulage accidents in coal mines can and will be reduced when (1) the management is thoroughly convinced that such accidents are unnecessary and preventable, (2) when the management is imbued with the firm determination that the accident rate must be kept to the irreducible minimum, and (3) when safety is made a major operating problem.

Possible failure to effect an appreciable reduction in accidents due to haulage should occasion neither despair over the future nor indifference to present opportunities for progress.

There is a definite responsibility resting alike upon employer and employee. The employer owes a duty to the employee to furnish the safest possible working environment, and the employee owes a duty to himself, to his fellow worker, and to his employer to perform his duties in the safest and most efficient manner possible.

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March, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFETY IN CONNECTION WITH HAULAGE PRACTICES IN ALABAMA COAL MINES[1]

By F. E. Cash[2]

INTRODUCTION

Although Alabama has shown an annual reduction in fatal accidents from all causes for the past four years, the trend for haulage accidents when considered separately is in the opposite direction. The fact that the same tendency exists in other, and probably many, mining States is poor consolation. Although the writer is not able to solve the problem and give definite ways and means of reducing these accidents, it is hoped that by describing systems, methods, and practices used and accidents occurring in Alabama, some data will be given or points will be brought out that will enable mine operators to profit by the experiences in Alabama.

NATURAL CONDITIONS

Coal is being mined in 29 beds which vary as to the thickness of partings and the nature of coal, roof, and floor. Moreover, in the same bed of coal in adjoining mines or even in the same mine, natural conditions that materially affect haulage are different.

The mines may be divided into four groups or classes: slope, drift, shaft, and strip mines. This does not mean that where a mine is classed as a drift or shaft mine there may not be inside slopes, nor that when classed as a slope mine the production may not come from undulating or level coal strata. There are mines in the State which require four couplings or handlings of cars between the face and the tipple. This might be from mule to locomotive to inside hoist, to locomotive, to outside hoist; or it may be from mule to hoist, to locomotive, to hoist, to some form of surface haulage.

The variable thickness of the coal beds and the large amount of rock partings make it productively necessary to gob as much rock as possible underground. In some mines the monthly production averages a car of rock hauled to the surface for each car of coal loaded. These natural conditions increase the haulage hazards over those in some mining districts.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6243."

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METHODS OF TRANSPORTATION

There are a number of methods and combinations of methods of transportation employed in the 220 operating coal mines of the State. Comparison will be based on the number of mines using the methods rather than on the tonnage produced. Combined mule and hoist haulage is most generally used; next comes locomotive and hoist; third, mule, locomotive, and hoist; fourth, the locomotive. A few mines use locomotives and belts or locomotives and endless rope hauls.

TYPES OF EQUIPMENT

Main or Outside Hoists

The hoisting engines are driven by air, steam, or electric power. These units may be directly connected or geared to a single or double drum hoist. Practically all of the hoists are equipped with two independent sets of brakes and an indicator showing continuously the rope location; some installations have over-winding devices and governors. The State has some modern up-to-date hoisting plants and some which are obsolescent types.

Haulage Signals

Hoisting signals are not uniform throughout the State. The code of signals is posted in view of the engineer and also at all underground stations. The signal wires are generally on a circuit of 32 volts, or less, and telephones are usually installed for special or emergency use. For operating locomotives and trips, some mines have automatic signals, others have hand-operated signals, and in some of the larger mines dispatchers control and regulate the movement of each trip in the mine.

Haulage Ropes

Haulage or hoisting ropes vary in size from ½ inch to 1½ inches for use under different mine conditions. At some mines the ropes are sheaved, run on rollers, greased regularly, inspected twice daily and written signed reports made of such inspections; at other mines the care of hoisting ropes is not nearly so commendable.

Cars and Skips

A few of the mines have underground pockets where the coal is unloaded from the mine car into a loading pocket, transferred to a skip or belt, and hoisted to the surface. These skips vary in capacity from 1½ to 9 tons, and in addition to the socketed rope-clevis coupling, a safety rope is passed around the skip and clamped to the hoisting rope ahead of the rope socket.

The different kinds and sizes of mine cars in use are numerous and vary in capacity from one-half to 3 tons with an average coal-carrying capacity of 1 ton. Some of the cars are steel, some wood, and some steel and wood.

Locomotives

All locomotives used underground are electric. A large majority operate from 250-volt direct current, but a few storage-battery locomotives are in service. Locomotives weigh 4 to 15 tons and are equipped with trolley pole, cable reel, or crab reel. In general practice a locomotive in operating condition must have a light at each end and be equipped with a bell. In some mines "backing" or "back-poling" on the trolley pole is allowed but it is generally prohibited.

Tracks

The gage varies in different mines from 24 to 48 inches and rail weights from 60 to 20 pounds per yard. The tracks have wooden ties principally on main haulage and wooden or steel ties for room work. There is at least one exception to this practice, in that one mine uses steel ties on main line as well as room tracks and switches. Most of the switch throws are at right angles to the track, but some are parallel; in some instances provision is made in the ribs for switch throws, so that the trip rider has additional protection if a trip wrecks in passing over a switch.

STATISTICS

The data from which the following tables have been prepared are from figures published and furnished by the Inspection Department of Alabama.

Table 1.- Total fatal accidents from all causes, in Alabama coal mines, 1925 to Oct. 31, 1929

Causes	1925	1926	1927	1928	1929 through October
Falls of rock and coal	49	42	61	28	24
Haulage	20	31	13	17	17
Electricity	11	18	9	13	9
Explosives	7	-	-	4	-
Machinery	2	-	2	3	-
Gas and dust explosions	66	42	3	-	11
Miscellaneous	7	6	5	2	-
Total	162	139	93	67	61

As the table shows, 17 of the 61 fatal accidents during the first 10 months of 1929 were haulage accidents. This figure, based on tons produced and man-days worked, is an increase over the entire year of 1928.

Table 2.- Haulage Fatality Statistics

Year	Operating mines	Tons produced	Average days operated	Total number men
1925	222	20,408,931	210	26,406
1926	222	21,508,812	235	26,647
1927	213	20,190,926	202	27,152
1928	221	18,056,116	170	25,396

Year	Man-days worked	Total haulage fatalities	Tons per haulage fatality	Man-days per haulage fatality
1925	5,545,260	20	1,020,447	277,263
1926	6,262,045	31	693,832	202,001
1927	5,484,704	13	1,553,148	421,900
1928	4,317,320	17	1,062,124	253,960

STATE LAWS

The State mining law governing riding cars and trips and traveling on slopes follows:

Sec. 117. Persons Permitted to Ride on Cars:- No person or persons except those in charge of trips, superintendents, mine foreman, electricians, machinists, and blacksmiths and others, when required by their duty, shall ride on haulage trips, except a special trip of empty cars may be operated for the purpose of taking employees into and out of the mine, when the distance to and from their work exceeds one mile. No persons, excepting trip riders, shall ride on loaded car or cars, and they shall ride only the front or rear end of the trip.

Sec. 118. Not allowed to Travel on Foot on Plane, etc.:- No person shall be allowed to travel on foot to or from his work on any incline plane, rope or locomotive roads, when other roads are provided for that purpose.

COMPANY RULES

Some of the outstanding rules used by the operating coal companies for the guidance of motormen and trip riders, are:

No trip rider shall ride between cars, either loaded or empty. When empties are being pushed ahead of motor, the trip rider must ride in the second car from the front, and with one trip rider,

when trips are being pulled, he must ride in the rear car, and with two trip riders, one should ride the front car and the other the rear car. No trip rider shall ride on the motor when there is a trip of cars either loaded or empty attached to the motor.

All motormen are required to wear shoes with rubber soles when on duty.

No employee is permitted to couple cars except designated motorman and trip riders, unless he is instructed to do so by the Superintendent, mine foreman or their assistants.

No motorman will be permitted to back up on the trolley pole except in entries where clearance is not sufficient to turn it, and under such conditions, motormen must run at reasonable speed and with motor under control until a place is reached where clearance is sufficient.

Motormen shall not pass a red signal light; also, if signal lights will not burn he must send his trip rider ahead at least 100 feet as a flag.

Motormen must not allow any one to ride on the locomotive except such as are specially authorized to do so.

Motormen are positively forbidden to run their locomotives without proper headlights. In case a head light burns out on a locomotive, said motorman must not leave his terminal again until the headlight has been put in repair. A light must be carried on each end of a trip.

At places where regular flagmen are employed, or where block signals are used, motor men must not run by these places without first getting a clear signal. NO SIGNAL shall mean the same as a signal to stop.

Pole holders are required to wear gloves at all times while on duty.

Motormen must not leave their motors unattended at any time.

The bucking of cars or motors on the track is positively prohibited. The presence of a Bucker on the motor or trip is a violation of the rules and both motorman and trip rider shall be subject to discipline.

All motors are to be equipped with re-railers and motormen are held personally responsible for same.

No trip rider or other employee shall ride on top of loaded cars or between the cars either empty or loaded. When empties are being pushed ahead of motor, the trip rider must ride in the second car from the front.

It shall be a violation of the rules of the mine to make running switches except at the main yard.

HAULAGE ACCIDENTS

A description of a few of the fatal, lost-time and no-lost-time accidents occurring during 1929 brings out graphically the salient points in connection with violation of safe haulage practices.

Shaft Accident

A white miner, age 50, was killed in main hoisting shaft June 7. The mine foreman and four other men were on the cage coming to the top. The mine foreman claimed he gave the regular signal of 3 bells, signifying a man trip. The engineer claims he got only one bell, which is the regular hoist signal. As the cage approached the top of the shaft the mine foreman alighted while the cage was in motion. The deceased attempted to get off, struck his head on a headframe beam and was knocked down the shaft. The shaft has safety gates, but the mine foreman opened the gates and got off safely, indicating the slow speed of the cage. Had no one attempted to alight at the shaft collar and stayed on the cage until it reached the regular coal landing, there would have been no accident. The mine foreman should not have alighted from the cage while in motion.

The following statement and table are quoted from the Alabama Holmes Bulletin 9, from which the foregoing description also was taken:

Eleven lives lost in six months on haulageways, more than doubles the number in the same period of 1928, all preventable, all due to some one's failure to do the right thing. Study the table below and see what carelessness, violation of safety rules and violation of the 'Rule of Common Sense' is costing.

Jumping on or off cars or hoist while in motion	4
Defective couplings	3
Ran into loaded trip	2
Fell under car	1
Failed to wait for signal	1
Total haulage fatalities	11

In the shaft accident previously described, the man should not have gotten off the moving cage. The example set by the foreman, who is responsible for the safe and efficient operation of the mine, is deplorable. The officials or those in charge of men must place more importance on preventing accidents and on observing rules themselves as well as

requiring others to observe them. This accident as described was unquestionably caused by an official's disregard of company rules.

Slope Accident

A miner, age 35, and two other men were standing between the slope track and a track going in a right entry. A trip of 6 empty cars on the chain was coming down the slope. The chainer signaled the engineer for this trip to enter first right. About 75 feet above the switch the cars became uncoupled, wrecking at the point where the deceased was standing, causing injuries which resulted in his death five hours later. Investigation developed a spread coupling.

Undoubtedly the men knew that they were not in a safe place, but from long familiarity with this particular haulage they disregarded the danger. To prevent the recurrence of a similar accident men should be prohibited from loitering on haulageways. Since inspection and upkeep are as essential for safe couplings as for safe hoisting rope, it is evident that this accident was due directly to defective equipment; therefore provision should be made to keep cars, couplings, pins, track, and other haulage accessories in a good state of repair.

Locomotive Accident

A trip rider, age 23, and a motorman were bringing a 'motor' and 12 loaded cars out of 8th east entry. The motorman was sitting on the front end of the 'motor' and deceased was riding rear end bumper with part of his body extended over the motor. For some reason, due most likely to the push of 12 cars, the 'motor' became derailed at the rear end at a point where the roof is only 4 feet above the rail, and as the motor stands 3 feet above the rail, this gives only 12 inches clearance at that point. Where track is intact and derailment occurs the wheels must rise high enough to permit flanges to mount rail. In this case the rear end of motor was raised enough to crush trip rider.

A trip rider should never ride or be allowed to ride on the front end or on top of a locomotive. The rules of the operating company in whose mine this accident occurred prohibit the trip rider from riding on the front end of the locomotive. Lack of sufficient or of efficient supervision may have played a part in the occurrence of this accident, but the disregard of company rules by the trip rider resulted in his death.

Trolley Wire Accident

Guarded Wire:

A coal loader and his two helpers were pushing a car on the heading opposite a room when the loader raised up coming in contact

with the trolley wire, which was 49 inches above the rail, and was killed. The shielding at this point had been knocked off previously by a trolley pole and it had not been replaced.

All trolley wire which is less than 6½ feet above the rail should be guarded, but to allow the guards to be broken down and not replaced gives false protection. As soon as this shielding was broken off the proper repair men should have been notified to repair it as soon as possible, and in the meantime men working in the section should have been cautioned that the shielding at a certain point was in bad order.

Trolley Wire Accident

Unguarded Wire:

A motorman stopped his locomotive on a switch, raised up and struck the back of his neck against the trolley wire. His trip rider pulled him away from the wire and gave artificial respiration but was unable to resuscitate him.

Had the trolley wire been guarded even over the switch, or if the motorman had been wearing insulated shoes, the accident in all probability would not have happened.

Mule Accident

A mule driver, age 32, was coming out with a loaded trip. There is a hill about 75 feet from sidetrack where all drivers should stop and wait for a signal from sidetrack before coming. This he failed to do. The driver ahead of him failed to get in the clear before deceased struck him. Investigation developed deceased was in front of trip trying to stop it when the butt stick of mule harness jammed his head between stick and car, puncturing his skull and causing his death. There was no derailment of cars. This is a rather unusual accident.

A driver should never ride on the front end of a car or trip.

Car Accident

A coal loader, age 30, was working in the aircourse, bringing his loaded cars to heading, riding the rear end of a car. There being a heavy grade he ran into a loaded trip of cars on heading, was thrown into gob and so injured that his death resulted.

The coal loader was taking a chance and he knew it. It was a part of his work to drop the car out to the heading so that the locomotive could get it, but the car was to be brought out only after determining that the way was clear and then at a reasonable speed controlled by sprags. This accident brings out the importance of safety education and the

necessity for close supervision. This case shows how the personal equation may at any time overturn even the best of supervision or education.

Trip Accident

A trip rider near the center of a 40-car trip of loads, raised up, striking his head against a trolley hanger or cross collar, which knocked him between the moving cars resulting in his death.

Riding in the center of a trip is a violation of the State mining law as well as of the operating company's safety rules and of ordinary common sense. Closer supervision and observance of mining laws and safety rules will prevent the recurrence of similar accidents.

Pushed Trip Accident

A trip of empty cars was being pushed by a locomotive when the front end of the front car split a switch, jumped the track, and caught the trip rider between the car and the rib, resulting in his death.

Wherever possible, trips should be pulled - and not pushed. Under no conditions should a coupler ride on the front end of a trip.

Lost-Time Accidents

Some accidents causing loss of time, as described by the company official investigating the accidents, are:

1.- A trip rider was rerailing or "bucking" a wrecked loaded rock car with a piece of rail; when the car started, his finger was caught between car and rail, resulting in the amputation of a finger.

This company provides rerailers on all locomotives. Use of the rerailer would have prevented this accident.

2.- A miner was switching cars into his room. Two cars were on the entry when he uncoupled one car and, as he thought, blocked the other car. When he had pushed the first car a short distance the second car ran down catching him between the cars fracturing a rib and injuring the chest.

Sprags were provided, but they were either not used or were improperly used.

3.- Injured man was pushing a loaded car under low top with his right hand placed on top of the car, when the hand was caught between top of car and roof, resulting in laceration of two fingers.

This is an old miner and should have known not to place hands on top or sides of cars. We have had several accidents of this kind recently and foremen should make a drive among all men against this unsafe practice.

The responsibility for the prevention of accidents similar to the three given rests with the foreman directly in charge of the work. By closer supervision, caution, and discipline he can prevent such accidents.

No-Lost-Time Accidents

Injured pushed empty car into working place and picked up a small rock to block it with. The car broke the rock and ran over and mashed a finger on his right hand.

Sprags were provided and the company rules required their use, but the men ignored the rule and the foreman failed to enforce it.

A trip rider was on a trip, and when he passed through a door his left hand was caught between the car and door, bruising the back of his hand.

This accident was due to the carelessness or thoughtlessness of the injured man.

SUMMARY

The accidents from the various causes described are typical of those occurring in connection with haulage. They are the result of negligence on the part of officials in charge, lack of supervision where dangers exist, failure to repair or replace faulty equipment, or of ignorance or disregard of State laws and mining rules by the foremen and men; least of all are they due to so-called trade risks or unavoidable causes.

Some of the specific causes and results of accidents on haulage are:

1. Improper blocking of cars, causing mashed fingers and crushed bodies.
2. Defective switches, causing derailments and various accidents.
3. Defective couplings, causing broken trips and running over men on haulageways.
4. Coupling cars while in motion, mashing fingers and hands.
5. Pushed locomotive trips, resulting in wrecks and injuries to trip riders.
6. Getting off and on cars or locomotives while trips are in motion.
7. Riding center or front end of trips and on top of locomotive.
8. Unguarded trolley wire in low coal.
9. Disregard for blocks and signals on haulage.
10. Operating trips without lights or markers on both ends.
11. Inadequate maintenance of wire, locomotives, and track.
12. Failure of officials to set a good example by using safe practices in haulage.
13. Failure of officials to discipline men found using unsafe methods, practices, or devices in haulage. Only too frequently an inherently unsafe practice is allowed to be used so long as nothing unfavorable occurs.

CONCLUSIONS

Some of the specific things which men and officials can do to prevent haulage accidents are:-

1. Start the new man to work with the proper knowledge of his work and its hazards.

2. Supervise and maintain strict safety discipline at all times.

3. By inspection and proper care, see that all equipment is kept in safe operating condition.

4. Study and become well acquainted with dangers, unsafe practices, and conditions and do what is possible to correct or guard against them.

5. Put safety on the same business basis as production.

In order to lessen the number, decrease the severity, and eventually prevent the numerous haulage as well as other accidents, every man connected with the mine should assume part of the responsibility. First, the operating company and its department heads must realize that the prevention of accidents is a big operating problem. They must believe in safety and require the officials and foremen under them to assume the same attitude. The officials and foremen must obey rules and laws and force the men under them, by education, supervision, and discipline to observe the rules and do their work in a safe manner. The men as individuals owe it to themselves, their families, their fellow men, as well as to the operating company, to know the safety rules and State mining laws, abide by them, and try to see that fellow workers have the proper regard for the safe operation of the mine. If any worker knows of definitely unsafe methods or practices in or around the mine it is his manifest duty to report such unsafe condition to officials in charge of the mine or to State inspection officials, if necessary.

I. C. 6244 March 1930

DEPARTMENT OF COMMERCE
UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

PRACTICAL APPLICATION AND COST OF PROPER ROCK-DUSTING

BY

G. M. KINTZ

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March, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

PRACTICAL APPLICATION AND COST OF PROPER ROCK-DUSTING[1]

By G. M. Kintz[2]

Methods and costs of rock-dusting bituminous coal mines have been discussed in various publications by the United States Bureau of Mines and others.[3] This paper discusses the present methods and cost of dusting at the Swastika mine of the St. Louis, Rocky Mountain and Pacific Co., Raton, N. M.

In the Swastika mine the recommendations of the United States Bureau of Mines with regard to rock-dusting are followed at least as well as in any mine in the States of Wyoming, Colorado, and New Mexico. This mine is located at Swastika, N. M., approximately 4 miles west of the town of Raton and is in the Raton coal seam. The mine is rated as gassy and produces approximately 1,000 tons of coal per day.

A mining system known as the "room-and-pillar with cut-offs" is used which results in recovery of nearly 100 per cent of the coal. The mine employs approximately 250 men.

Ventilation is maintained by a Sirocco fan so arranged that the air current may be reversed and exhausting approximately 130,000 cubic feet of air per minute at 3 inches water gauge. This fan is supplemented by two auxiliary fans for emergency use. Three main entries penetrate the hill in a straight line for approximately 8,000 feet. The haulage is through the center entry, which is also used as the intake in conjunction with a two-compartment concrete-lined shaft located at the end of the entries. The main return for 3,000 feet inby the fan is 12 feet wide and 7 feet high. For this distance the ribs have been reinforced by a wall made of coke-breeze blocks plastered over with cement. Solid concrete blocks are used for the top row of blocks of the wall. Steel 10-inch I-beams spaced 40 inches apart center to center are supported on these solid blocks and thus support the roof. The next 1,000 feet inby has the same area, but the cement blocks extend to the roof, as no crosspieces are needed. The airways are kept clean and well rock-dusted.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6244."

2 - Associate mining engineer, U. S. Bureau of Mines.

3 - Rice, G. S., Stone-Dusting or Rock-Dusting to Prevent Coal-Dust Explosions. Bull. 225, Bureau of Mines, 1924, 57 pp.

Rice, G. S., Safety in Coal Mining (A Handbook). Bull. 277, Bureau of Mines, 1928, pp. 54-58.

Owings, C. W., and Dodge, C. H., Methods and Costs of Rock-Dusting Bituminous Coal Mines, Bull. 18, Carnegie Institute of Technology, 1925, 192 pp.

A rather unique system is used to handle gas feeders along the intake and haulage entries. Where these feeders occur in the roof or where gas is apt to collect in high points along the haulage, tin funnels as large as 3 feet in diameter are placed near the roof. From these a 2-inch or other suitable-sized pipe is run into the return through the wall of the nearest break-through. A valve on this pipe line controls the amount of air pulled. These devices keep the high places clear of any gas, even when strong feeders are encountered.

Fifteen-ton electric trolley locomotives are used on the main haulage and approximately 30 roller-bearing cars, each having a capacity of approximately 3,000 pounds of coal, compose a trip. All loaded trips carry steel drags on the last car, and automatic derails are located at advantageous points throughout the mine.

SPRINKLING

All loaded trips are thoroughly sprinkled on the sectional partings and at the main parting. The track is sprinkled every idle day, or if needed, every day, from a large tank mounted on a truck hauled by a motor. Neither nozzle nor spray is used, as these were unsatisfactory. The water leaves the tank through a 1½-inch valve and is permitted to flow on the road through a horizontal 1½-inch T. Water is obtained from filling stations located throughout the mine. It is stated by the officials that the use of this method of sprinkling permits the entire haulage roadbed to be wetted down in two hours.

No sprinkling is done at the working faces and no water is used on the cutter bar of the two mining machines used in a section of the mine under a creek bed where the coal contains a considerable quantity of water and no gas has ever been encountered. The cost of the installation and maintenance of sufficient pipe lines thoroughly to wet down the coal-dust at the working faces would probably far exceed the present cost of rock-dusting and require the same supervision.

If a sprinkling system is used, a force of men must be kept employed wetting down abandoned regions, whereas proper rock-dusting permits abandonment of entries and rooms without further consideration of the dangers of dust explosions, if due regard is given to ventilation.

ROCK-DUSTING

James R. Barber, assistant engineer, St. Louis Rocky Mountain and Pacific Co., Raton, N. M., in an article in the Mining Congress Journal of February, 1928, volume 14, No. 2, page 90, gives a fairly complete description of the rock-dusting methods used in the mine at that time; from this article the following paragraphs are taken:

> Our first experiments with rock-dust were made in 1924, using crushed shale, which proved unsatisfactory. A ball mill was then installed, using purchased limestone. This practice was abandoned on account of the ball mill being unsuited to the size of limestone we could secure.

We have, for several years now, been dusting with ground limestone, shipped sacked, prepared by the Colorado Portland Cement Co. to a fineness such that 60 per cent is minus 200 mesh; all minus 60 mesh. This material is very satisfactory for our purpose.

The method of dusting for main or cross entries is to proceed to the face with the train and dust outward against the intake air. The machine, when operating, makes a dense cloud of white dust; but by backing against the air little discomfort to breathing is experienced. One mile of entry can be dusted in about an hour.

For air courses containing no tracks or trolley wire, we have to employ a modified method. Six-inch holes are cut in crosscut stoppings at about 500-ft. intervals. Into this hole a 6-inch galvanized iron pipe is inserted and cemented in. In the end of this pipe toward the haulageway a 1/4-inch plate of iron is bolted on with four bolts, a rubber gasket being placed behind the iron plate. This makes an air-tight job. When the back entry is to be dusted, the machine is stopped at one of these pipe stoppings, the iron plate is removed, the blower pipe is inserted through the stopping pipe and dust is blown into the back entry for a half-hour period. The return air in the back entry picks up the dust as it is blown in and deposits it fairly well over roof, sides, and bottom. Supplementing the blowing of dust through stoppings in back entries, we use hand-dusting.

The standard of incombustible content on rib, roof, and bottom is purposely kept high (70 per cent), and in order to maintain this standard dusting is done lightly almost every week.

Dusting tests are made at regular intervals. Samples are taken 6 inches wide of roof, rib, and bottom (not between rails), 7 samples to a mine. These samples are taken to the laboratory, screened through a 20-mesh screen, and the incombustible content ascertained by means of a volumeter. This method is simple, accurate, and rapid. A sample's content can be determined in about 10 minutes. The results of the tests are then entered in the dust-testing book, and typewritten copies furnished to president, general manager, and superintendents.

As showing the high content of incombustible found, a series of recent tests is given:

	Per cent incombustible
Main, at 4th south.....	88
Main, at big parting....	80
3rd east at 3rd north..	78
3rd south at 2nd east..	73
1st east off 4th south	80

> At each pump we place an open nail keg of rock-dust to be scattered by hand in case a fire is started by any cause, such as a short circuit or overhead motor.
>
> Rock-dusting greatly improves the lighting in mines.
>
> We are thoroughly satisfied with rock-dusting and believe we are protected from any large catastrophe. The cost of continuous rock-dusting is somewhat less than 1 cent per ton of production.

Soon after this article was published the practices discussed by Mr. Barber were modified and the company took the added precaution of requiring the miner to rock-dust his working place to the face. The rock-dust is delivered in 80-pound sacks, with the miners' timber.

Most of the men regard rock-dusting as being in the same safety category as timbering; and like timbering, rock-dusting requires strict supervision to be kept up to a definite standard. Five months after the system was inaugurated the miners were dusting to the face. The management then decided that perhaps 10 feet from the face would be sufficiently close. The results of this plan were apparent when, on a visit to the various working faces a year later, it was observed that in only one or two places was the dust being kept to within 10 feet of the face and in some rooms and entries the dust was as far as 60 feet back. The officials have now returned to their original plan, and the miners must keep the dust up to the working face.

The American Engineering Standards Committee in cooperation with the United States Bureau of Mines made the following recommendation in Bureau of Mines Information Circular 6030:

> 5. Parts of Mine to be Used: Rock-dust shall be distributed on all main haulages, all entries to the last break-through, in all rooms and pillar workings, to within 40 feet of the face, or to the last break-through, and in all return airways where hauling or travelling is done. Dust must be distributed upon top, bottom, and sides of places.

When the "diggers" are required to dust their working places, a far different problem is presented than when the company employs men to do the dusting. The question which naturally arises in most operators' minds is: What difficulties were encountered by the company in getting the men to do the rock-dusting at or near the faces?

The men employed at the Swastika mine are American, Italian, Negro, Slavish, and Mexican, their numbers corresponding roughly with this order. When informed that they would be expected to keep their places rock-dusted without additional compensation only one of the "diggers" raised any objection. This is probably due to the fact that prior to issuance of the order, a moving picture was shown illustrating the dangers of an explosion in a mine

that was not thoroughly rock-dusted and how rock-dust prevented coal-dust explosions. Besides this the miners appear to have high regard for the opinions of their officials and cooperate with them in keeping the mine safe.

In the pillar workings a heavy deposit of rock-dust is maintained by the men on the fallen roof rock at the edge of the caved area and on the floor back of the pillar face. Should an explosion occur adjacent to a caved area a large quantity of rock-dust would be thrown into the air at the source of the ignition. There is no doubt that this dust would tend definitely to prevent propagation of an explosion by coal-dust.

Some coal operators state that rock-dust on the coal retards its sale on the domestic market. The St. Louis, Rocky Mountain and Pacific Co. sells more than one-third of its total Swastika output to the domestic trade. In the two years that this system of dustin has been in force no complaints have been received with regard to rock-dust in or on the coal. The effect of rock-dust on the ash content of the mined coal may be estimated by considering a typical example. Consider a thoroughly rock-dusted rib of a room in a coal bed 6 feet thick. The average depth of undercut is 6 feet, hence the cross sectional area of the cut would be 36 square feet. It has been found that a well rock-dusted passageway in a 6-foot coal bed contains dust at the rate of 4 pounds per linear foot and if all of the dust were applied to the ribs, one rib would contain 2 pounds of rock-dust per linear foot. Therefore in each linear foot of rib taken from the cut to the full depth of 6 feet and the height of the bed, there would be 2 pounds of rock-dust to 36 cubic feet of coal, which at the rate of 80 pounds per cubic foot is equivalent to 2,880 pounds; consequently, .07 per cent ash would be added to the mined coal. If the rock-dust became mixed with the coal mined from the first 2 feet of depth of the cut, the added ash would be 0.2 per cent. Actually much of the rock-dust is thrown into the air and falls to the floor when the coal rib is shot, hence the effect of the rock-dust on the ash in the coal is negligible.

As previously stated, at the Swastika mine the trips are thoroughly sprinkled on leaving the sectional partings and again on the main parting. This naturally tends to wash off part of the dust deposited on the coal. Where this system of dusting up to the face has been used, very little rock-dust is observed on the coal at the tipple.

The cost of this system of rock-dusting is the next and most interesting question raised by the operators.

The following tables[4] give the amount of dust used and the cost of the dust and its application. The first table covers a period of years; the second shows cost and amount of dust used during the 14-month period that the men have been required to dust their own working places. It is noticeable that although the cost of rock-dusting was almost 2 cents per ton of coal produced over the entire period from August, 1928, to September, 1929, inclusive, during which time the men rock-dusted the face region and the company maintained the rock-dust in the regions back of the face, yet during the six months of 1929 for which records are available (April to September, inclusive) the total cost of the rock dusting was less than 1 cent per ton (and in two of these months was less than one-half cent per ton), the tonnage in these six months being approximately normal. This again establishes the fact that adequate rock-dusting can be kept current for 1 cent or less per ton.

4 - Figures furnished by and published with the permission of the company officials.

Table 1.- Cost of Rock-Dusting the Swastika Mine, August, 1928, to September, 1929, inclusive

Date	Cost of dust	Cost of labor	Sacks of dust	Tons of coal produced	Cost of dust per ton of coal
Aug., 1928	$ 339.59	$ 145.30	850	17,865	.0271
Sept.,1928	297.60	101.05	775	16,447	.02423
Oct., 1928	76.80	122.05	200	19,856	.01001
Nov., 1928	443.01	77.50	1,047	22,451	.02318
Dec., 1928	491.95	181.35	1,250	19,833	.03390
Jan., 1929	422.40	175.80	1,110	22,427	.02667
Feb., 1929	729.60	300.25	1,900	17,604	.0585
Mar., 1929	192.00	81.90	500	15,654	.0175
Apr., 1929	117.15	30.10	250	16.210	.00908
May, 1929	67.20	40.20	175	13,589	.007904
June, 1929	28.80	25.95	75	11,980	.00457
July, 1929	96.00	63.40	250	15,583	.01023
Aug., 1929	96.00	37.20	250	18,382	.007247
Sept.,1929	38.40	22.25	100	18,434	.00329
Total	$3,436.59	$1,404.30	8,732	246,315	.01965

Table 2.- Cost of Dusting Per Ton of Coal Produced

	Dust	Labor	Total cost
14-month period, Aug., 1928 - Sept., 1929	$0.01395	$0.00570	$0.01965
12-month period, Oct., 1928 - Sept., 1929	$0.01320	$0.00546	0.01866

Table 3.- Amount and Cost of Dust Distributed at the Swastika Mine, August, 1924 - June, 1929

Year	Pounds of lime-stone dust	Cost
Aug. - Dec., 1924	751,520	$1,350.05
1925	28,000	107.22
1926	234,400	892,47
1927	584,700	2,243.02
1928	781,700	3,000.23
Jan. - June, 1929	500,000	1,920.00
	2,880,320	9,512.99
Less amount shipped to other mines and on hand	152,210	272.44
Total	2,728.110	$9,240.55

Note: Total tonnage for period August, 1924 to June, 1929 was 1,108,725 or 2.5 - pounds of rock-dust per ton of coal produced.
$9,240.55 ÷ 1,108,725 = $0.0083 cost of rock-dust per ton of coal produced.

CONCLUSIONS

1. Rock-dusting can successfully be kept up to the working face.

2. Miners will cooperate and keep the working places rock-dusted without additional compensation.

3. Coal taken from a rock-dusted mine can successfully compete with any other coal in the domestic market without additional treatment to remove the rock-dust.

4. The amount of rock-dust in a ton of coal taken from this mine is less than one-fifth of 1 per cent by weight.

5. Rock-dusting of the caved areas around pillar workings will assist greatly in preventing the propagation of an explosion initiating in that area.

6. Rock-dusting permits the complete abandonment of entries or rooms at any time without requiring men to return to that area periodically, as would be the case where sprinkling is employed.

7. Where rock-dusting is properly done it is more permanent and decidedly cheaper than sprinkling if the sprinkling is done effectively.

8. The cost of properly rock-dusting the Swastika mine, including rooms, was kept at approximately 2 cents per ton of coal produced for a 14-month period, and during the last six months of that period the cost was less than 1 cent per ton of coal produced.

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FEBRUARY, 1930

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UNITED STATES BUREAU OF MINES
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EFFECT OF ABNORMAL AIR CONDITIONS ON MINE WORKERS

BY

R. R. SAYERS

L. C. 8245

FEBRUARY, 1950

I. C. 6245
February, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

EFFECT OF ABNORMAL AIR CONDITIONS ON MINE WORKERS[1]

By R. R. Sayers[2]

INTRODUCTION

The number of investigations reported during the past year indicates that control of the health hazard due to abnormal air conditions is still a matter of serious consideration in the mining, metallurgical, and allied industries.

SUMMARY OF RECENT INVESTIGATIONS ON SILICOSIS

Australia. - In an investigation into the health and working conditions of employees in the mining industry of Tasmania and Victoria[3], it was found that although pulmonary tuberculosis may occur early in life, silicosis is a disease of slow evolution and does not appear frequently under the age of 40. A period of at least ten years in the industry was found necessary to produce silicosis, and in no case was the disease found in those who had not worked underground. A very high incidence of silicosis was found in all fields among those who had worked underground 20 years and longer. There was some evidence that silicosis complicated by tuberculosis may occur earlier than uncomplicated silicosis.

England. - In a paper presented before the Section on Occupational Diseases of the British Medical Association, Dr. E. L. Middleton, the medical inspector of factories, called attention[4] to the occurrence of silicosis in various industries throughout the country.

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1 Report of the chairman of the Subcommittee on Physiological Factors, of the Committee on Ventilation of the Am. Inst. of Min. & Met. Eng., presented at the meeting in New York City, February, 1930. The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6245."

2 Chief surgeon, U. S. Bureau of Mines; surgeon, U. S. Public Health Service.

3 Moore, K. R. Report of an Investigation into the Health and Working Conditions of employees in the mining industry of Victoria and Tasmania, 1928. Service Publ. No. 8, Div. of Ind. Hyg., Commonwealth Dept. of Health of Australia. 29 pp.

4 British Medical Association, Section of Occupational Diseases. The Present Position of Silicosis in Britain. The Lancet, 217, No. 5528, Aug. 10, 1929, 282-283.

According to Dr. Middleton the sandstone industry is the most widespread of all silicosis-producing industries, but the incidence of the disease is diminishing owing to the application of water and mechanical methods of dust suppression. The granite industry, although allied to sandstone quarrying and dressing, is complicated because of the varying proportions of quartz, felspar, and mica present in the rock. As the result of an inquiry into the slate industry there appears to be some evidence of the production of fibrosis of the lungs, but the occurrence of actual disability within the ordinary working lifetime of workers is not clearly shown. In the grinding of metals silicosis, and especially the combined disease of silicosis with tuberculosis, was at one time extremely prevalent, but the sandstone wheel is being largely replaced by wheels of abrasives such as carborundum. Allied to the grinding of metals is the process of sand blasting of metals and the etching of glass, which consists in playing a jet of sand under high pressure against a metal or glass surface to produce certain effects. The process is becoming more extensively used, and it is a matter for serious consideration whether a substitute such as iron or steel grit could not be used to replace the siliceous grit as the abrading material. In pottery manufacture, both in the manufacture of earthenware and of china, Dr. Middleton suggested that "potter's asthma" is due to dust containing ground flint. Ground silica is used in many industries; and in the crushing and grinding of silica and in the industries in which ground silica is used silicosis sometimes of an acute type occurs. The evolution of dust in these industries is controlled by the use of mechanical methods of handling, inclosure of machinery, and the use of localized exhaust draught. Tin mining in Cornwall has had a high mortality rate from tuberculosis, and it has been shown that this is associated with silicosis. Drilling of the highly siliceous rock in which the tin ore occurs is now done through hollow steel with a water feed. Silicosis also occurs in coal miners employed on hard rock headings.

The report also states that the ganister and silica bricks manufactured in the refractories industry contain over 90 per cent of silica and that a medical board composed of specialists periodically examines each of the two or three thousand persons employed in this industry.

In a paper[5] read before the same meeting, Dr. P. Heffernan, tuberculosis officer in Derbyshire, referred to a hypothesis, advanced by himself and A. F. Green[6] about a year ago, that pulmonary silicosis is brought about by the colloidal activity of fresh silica hydrosol derived from the hydration of quartz dust in the lungs, and not by any poisoning of the cell protoplasm by soluble silica. He said that to some this distinction between a chemical poisoning and a physical adsorption seemed to be rather of academic than of practical interest but to his mind it seemed to be of fundamental and practical importance. One of his reasons for this belief is that the colloidal theory of silicosis affords an explanation of the inhibitory or delaying influence which certain substances, and the accelerative effect which certain other substances, appear to exercise on the development of silicosis, which influences are otherwise inexplicable.

5 Heffernan, P. Some notes on the Biophysics of Silica and the Etiology of Silicosis, Brit. Med. Jr. No. 3584, Sept. 14, 1929, 489 - 492.

6 Heffernan, P., and Green, A. T. The Method of Action of Silica Dust in the Lungs. Jour. Indust. Hyg., 10, No. 8, 1928.

In an editorial appearing in the same issue with the papers by Middleton and Heffernan, the British Medical Journal[7] has the following comment on these two papers:

> A good example of the pathological change brought about by dust inhalation was contributed by Dr. P. Heffernan, tuberculosis officer in Derbyshire, in a paper which he read before the Section of Occupational Diseases at the Annual Meeting of the British Medical Association at Manchester; the paper is published in this issue at page 489. According to King's table, which is cited by Dr. Heffernan, a normal lung contains 1.40 mg. of silica per gram of the organ, and a silicotic one 123.55 mg.; silica is thus a normal constituent of the human body, and in no sense a poisonous substance. In order to operate actively it must enter into the intimate structure of the cell, and Heffernan regards the resulting fibrosis as being of a physico-chemical nature, due to the local action of hydrated silica upon the pulmonary tissue. There is some evidence, moreover, that certain substances - alkalis for instance - favour the formation of silica hydrosol from silica, accelerating the fibrotic development; others - for example, coal dust and clays - retard or inhibit this process. In this way an explanation is afforded of the very few cases observed of so-called "acute" silicosis (developing within two or three years of constant exposure, and then following a rapid course) from inhalation of a mixture of silica dust and strongly alkaline dried soap. In a similar way the absence of disabling silicosis among coal miners and fireclay workers can be explained. Gye and Kettle's experiments have led to the belief that the inhaled silica particles undergo slow hydration into silica sol. One might go further and say, the more soluble the form of silica becomes (hydrated, amorphous, crystaline) the more readily will it set up fibrosis.
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> All observers in South Africa and in this country are now agreed, as Dr. E. L. Middleton states in his paper, published to-day at page 485, that the phagocytes, when overloaded with the dust, lose their motile power, collect in the lymphatic channels, and lead to a reaction, with formation of fibroblasts, in the surrounding tissues. Here in this colloidal reactivity of the cell we are in the presence, as Heffernan suggests, of something analogous to the mysterious change in the vital behaviour of a cell when it takes on malignant growth.
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> These theoretical considerations, though interesting, do not give much assistance in solving the practical problem how to control the malady. The more widely radiological examinations are made of the lungs of workmen, in no matter what dustry occupation - coal miners, millers, cement workers - the more generally is some fibrosis found to be present in all. Only exceptionally, however, as in the case of the absorption of asbestos dust, is the process likely to develop so as to interfere with the normal working capacity and with life. Some standard, therefore, of what constitutes an injurious amount of silica dust is demanded, or legislation will become

7 Same as Ref. 5 above, page 504.

punitive on some quite harmless industries. Thus Middleton says: "Where the amount of dust is small the action of the phagocytes is likely to be effective, but where it is great . . ." What dosage is indicated by "small" and "great" in this connexion? If the meaning of "small" were known, would it be correct to go on to say, as he does, "each successive dose of silica produces a summation of effects"? Then, again, would it be fair to say "a comparatively brief period of exposure may be the only determining factor"? What, in minutes, hours, or days per week, per month, or year, is intended by "a comparatively brief period"? "Intermittency of the process" is another expression used in summing up the reasons for and against a standardization of air purity by a gravimetric process or enumeration of the particles; the arguments against such a standardization are, by Middleton, considered strong, but they are by no means convincing to us. A standard has been accepted in respect of the amount of lead in milligrams per 10 cubic metres of air; Dr. Ronald Lane showed in another paper read that adherence to this makes the occurrence of poisoning highly improbable. Mavrogordato has placed on record figures of silica dust at different stages of the improvements in the workings of the Rand mines, and such a control is in constant operation in them today.

We emphasize these points because apparently the Government has taken the unusual course of legislating first and postponing inquiry to a later date, as in the "Various Industries (Silicosis) Scheme, 1929." The insurance companies also have a considerable interest in the matter of compensation for the silicosis risk. Where exposure - as in the refractories industries, the grinding industries, sandstone industry, and pottery industry - is day in and day out, they rightly demand a premium far higher than for any other disease scheduled under the Workmen's Compensation Act - 5 per cent on the wages bill. Now every industrial employer, such as the paint and varnish manufacturer, who has a sack of crushed silica in his store room, from which he extracts once a week enough to make a paste with spirit and add to his other ingredients in order to give them the necessary "bite," becomes involved, and may have to abandon the use of the material or pay 8 per cent on his wages bill for an absolutely negligible risk. For meting out justice standardization seems urgently called for.

According to Stewart[8] in districts of Lancashire County, England, where the industry of coal mining predominates (40 per cent of the occupied males being engaged in mining) the male death rate from pulmonary tuberculosis is higher than the corresponding rate for the whole administrative county. Since the death rate from pulmonary tuberculosis among all coal miners of England and Wales is below that for the general population, the statistical evidence suggests strongly that there is in Lancashire an occupational factor at work that raises the death rate, of miners, and that factor is silicosis.

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8 Stewart, J. Logan, Silicosis and Tuberculosis. The British Journal of Tuberculosis, vol. 23, No. 1, January, 1929. 6-11.

An inquiry was made by Sutherland and Bryson[9] to obtain evidence regarding the occurrence of silicosis among workers in England manipulating sandstone in connection with the granting of compensation to sandstone workers since February 1, 1929, under the Workmen's Compensation Act of 1925. The findings disclosed that one out of every four men at work appeared to be the subject of "masons's disease." Since the men were all examined at work, any case so far advanced as third stage, and also many in the second stage, would not be at work.

Some of the provisions of the new Sandstone Industry (Silicosis) Scheme[10], which became effective in February, 1929, are that a register, supplied by the employer or the Medical Board, on which shall be entered date of commencement of his employment with employer, particulars of any previous employment in the sandstone industry or in any of the occupations mentioned in the First Schedule to this Scheme, and date of medical examination entered by the Medical Officer making the examination. The workman will be liable to forfeit his right to compensation if he fails to furnish true information or carry out other specified duties under this scheme. A workman employed in the industry on or after April 1, 1929, or in case of death, his dependants, will be entitled to compensation, if death or disability is caused by silicosis. The disease must be due to employment in the industry, and will be so considered if the workman has been employed for a period or periods amounting to not less than five years, unless the employer or the company proves the contrary. If the employer fails to carry out any of his duties he will be liable to pay an increased subscription to the Compensation Fund. The Scheme requires the employers to furnish facilities for the periodic medical examination made at the works and not, without the consent in writing of the Medical Board, to engage or to continue to employ in the sandstone industry any workman who has been suspended under the Scheme or who has refused to submit himself to any examination required under the Scheme. If the workman refuses or wilfully neglects to submit himself to any such examination or in any way obstructs the same, or if, after having been suspended and without consent in writing of the Medical Board, he re-engages in the sandstone industry, he will be liable to forfeit his right to compensation.

France. - In France a histochemical study was made of lungs taken from subjects having lived in the city and never having worked in coal mines[11] and in another study by the same investigators[12] a cytological and histochemical examination was made of the lungs showing four typical causes of anthracosis in miners in Courriere. According to these investigators, their studies have enabled them to contribute some new data in regard to the much discussed problem of the nature of anthracosis pigment, whether the ferruginous pigment found is derived from blood or from masses of exogenous particles of coal introduced by respiration.

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9 Sutherland, C. L., and Bryson, S. Report on the Occurrence of Silicosis Among Sandstone Workers. London, 1929, 41 pp.

10 Same as reference 9.

11 Policard, A., Doubrow, S., and Pillet, D. Reserches histochimiques sur l'anthracose pulmonaire. Compt. rend. de l'Académie des Sciences, vol. 188, No. 3, January 14, 1929, 278-279.

12 Reserches histochimiques sur l'anthracose pulmonaire des mineurs, by the same authors as ref. 11. Compt. rend. de la Société de Biologie, No. 6, February 15, 1929, 400-401.

From these studies the following conclusions were drawn: A small part of the pigment encountered in the anthracosis of miners is certainly formed by a substance derived from hemoglobin, yielding red ashes of iron oxide. The other part, and by far the more important, is formed by a substance which disappears during incineration without leaving any residue of iron oxide. This pigment, which is extremely abundant, can be considered either as a non-ferruginous endogenous pigment or as from coal. The authors think that the latter interpretation is correct for the two following reasons: The particles of this pigment are extremely irregular and non-spherical; they co-exist always with other particles of a mineral nature, which on incineration leave an abundant residue of calcarious and silicious ashes. This constant existence of silicious minerals and black pigment indicates that this latter is coal. The coal dust is fixed in the lungs exactly as the silicious dust.

According to the authors, one finds, therefore, in the lung of the miner, but with a very great intensity and clearness, the same phenomenon that has been pointed out in the anthracosis lung of the non-miner, that is to say the co-existence of particles of coal and of silicic calcerous particles. The one never comes without the other. The points where the first accumulate are exactly the same as those where the second accumulate. From the histochemical point of view, all true anthracosis (from coal) is at the same time a silicosis.

Germany. - In Germany, a number of studies has been made in the metal polishing, sandblasting, and porcelain industries. Teleky and his coworkers[13] found that, in spite of outstanding improvement in conditions, the mortality from tuberculosis is very high among metal polishers, higher than among mining groups. They consider sandstone to be the most dangerous grinding material and recommend the substitution, if possible, of artificial stone and the introduction of other protective measures such as dust removal, reduction of working time, separate rooms for polishing, and medical supervision.

According to the German Association for Industrial Hygiene[14] which has been investigating the sandblasting industry and has published some of the results, the work "sandblasting" is involuntarily associated with ideas like dust, health hazard, and lung diseases. Therefore, a summary of the known methods followed and at the same time directions for removing the dust and for the protection of workers engaged in sandblasting are welcome. It is worthy of note that there is a great deal of apparatus that works free of dust even for work which can be done only with a free jet on account of the impracticability of the apparatus. Although statistics of injury to health through sandblasting are not available, it can be said that at least there is no increase in the hazard from industrial dust due to the introduction of sandblasting. When it is considered that previously with the use of simple aids, such as wire brushes, scrapers, etc., for scouring or for removing colored paints, etc., the worker was exposed to a far greater dust hazard,

13 Teleky, Rosenthal-Deussen, and Derdack. Staubgefährdung und Staubschädigungen der Metallschleifer insbesondere des bergischen Landes. Abstracted in Reichsarbeitsblatt, vol. 8, N.S. No. 32, page 220, Berlin, Nov. 15, 1928.

14 Das Sandstrahlgebläse unter besonderer Berücksichtigung der Massnahmen zur Vermeidung von Schädigungen seiner Verwendung. Same as ref. 13.

one can calmly declare that sandblasting has brought an improvement from a hygienic standpoint. The appropriate classification of the material and study of numerous samples are to be emphasized. Competent supervision would be of valuable assistance in carrying out preventive measures, also it would be of great value in plants that use sandblasting.

At the suggestion of the county industrial physician, the Tuberculosis Hospital at Schwetzingen[15] examined 23 of the men engaged in sandblasting in a metal polishing plant after it has been observed that several of the workers were affected by lung disease. Of the sandblasters examined all except two complained of troubles. This report emphasizes the finding that the pneumoconiosis caused by sand dust is more dangerous than anthracosis. The time of activity of these men in this industry was comparatively short - with one worker 8 years, with one 6 years, and with all the rest under 3 years. In most of the cases there was a pronounced roentgenological finding and in some changes of great severity and extent were observed. More severe pneumocotic lung changes were found with proportionately shorter activity not only among older workers but also among the younger ones, although X-ray examination revealed more severe changes in the older people than in the younger when the time of activity in the industry was the same. Thus, the average of the workers, examined by Müller, who had more severe lung changes, was 45 years, while the average age of the other workers was 32.6.

Kaestle[16] found, in an investigation of 500 porcelain workers in Germany, that the condition of the individual porcelain dust particles as to size, form, and hardness, their combination, solubility, and floating ability are of significance for their action on the body after being breathed. Pure burned porcelain meal, a glazed silicate, contains many sharply angular, pointed, cornered particles and is aggressive, while porcelain dust with more crumbly granules appears less harmful. In porcelain factories both kinds of dust are found in various mixtures. Pneumoconiosis appears to owe its existence principally to the presence of silicic acid in the inhaled dust. The crystalline form of SiO_2 is especially injurious. The majority of the porcelain workers inhale the above characterized porcelain dust in the manufacture of the raw materials and handling of the finished porcelain stock. In the case of older workers with relatively short working time the changes are clearer and more stubborn than in younger workers of similar working time. Here the vitality of the whole man and of the respiratory system play an essential role in the prevention of damage and the self-purification of the intruded dust.

The complaints of the older porcelain pneumocotics and the hazard to all workers with pneumoconiosis from acute and, without doubt still more tuberculous infection, justifies the initiation of definite demands which have been formulated by Kaestle as follows:

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15 v. Müller, Ernestine. Pneumokoniose bei Arbeitern eines Sandgebläses. Zentralblatt für Gewerbehygiene und Unfallverhütung, vol. 15, N.S. No. 5, May, 1928, 148-150.

16 Kaestle. Ueber die Porcellanlunge. Fortschritte auf dem Gebiete der Röntgenstrahlen, vol. 37, 369-380, Liepzig, 1928.

1. The young are to be kept out of the occupation until they reach a certain bodily maturity, since with not too great youth the self-purifying power of the lungs is greater than later. Dust injuries are possible to unlimited extent, and cause and dispose more or less to injuries of the respiratory organs and to danger from bacteria. Pneumonia in the case of dust-injured lungs, with their contraction of the breathing surfaces and other injuries to the tissues, may be especially dangerous.

2. Patients plainly affected by tuberculosis are to be separated from the porcelain industry, on account of possibility of hazard especially to young co-workers.

3. All porcelain workers are to be examined at regular intervals by competent physicians. Sick and dust endangered workers are to be rejected.

4. The later cases of pneumokoniosis and their sequelae are to be considered and handled as accidents in the meaning of the law.

According to Kaestle a porcelain factory will never become a health establishment for lungs.

Russia. - Since silica dust, especially SiO_2, is considered most dangerous as a cause of pneumoconiosis, it was expected a priori by investigators in Russia that a high percentage of the workers in the porcelain industry would have the disease, as porcelain dust contains large amounts of SiO_2. However, from an examination of 282 workers, in a porcelain factory near Moscow, all of whom had been employed in the industry for at least 15 years, this was not found to be the case,[17] as only 24.1 per cent showed suspicious indications of pneumoconiosis and 13.5 per cent were definitely affected by the disease.

South Africa. - A clear picture of conditions in South Africa, where the most extensive investigations on the subject of the dust hazard have been carried out, was presented by Dr. Mavrogordato and Dr. Irvine at the meeting of the Permanent International Commission for the Study of Industrial Diseases, held in Lyon, France, April 3-6, 1929.[18]

Dr. Mavrogordato summarized the current views on the etiology of silicosis under two headings: (1) cases of uncomplicated silicosis in which no infective element is present; and (2) cases of silicosis complicated by infective processes,

17 Holst, L., Kaplunova, D., and Santotsky, M. Die Pneumokoniose der Porzellanarbeitern im Röntgenbilde. Fortschritte auf dem Gebiete der Röntgenstrahlen, vol. 37, 358-368, Leipzig, 1928.

18 Mavrogordato, M. The Aetiology of Silicosis.
Irvine, L. G. The Diagnosis of Silicosis as an Occupational Disease.
Rapports IVe Réunion de la commission Internationale Permanente pour l'Etude des Malades Professionnelles, Lyon, 1929, 43-105.

of which the most common is tuberculosis. He said that dusts produced in certain industries are "phthisis-producing" and defined such a dust under four headings: (1) its nature; (2) size of particles; (3) faculty of being arrested in the tissues; and (4) concentration of dust, duration of exposure, and conditions of exposure.

Irvine stressed the importance of the infective factor in silicosis and emphasized the fact that it is uncommon to find a condition of clinically obvious tuberculosis present at the stage at which silicosis first becomes detectable. The very great majority of cases of silicosis found among working miners, when first detected, are from the clinical standpoint cases of uncomplicated or "simple" silicosis, without evidence of obvious or "overt" tuberculosis. Nevertheless, "simple" silicosis should not be thought of merely as a dust fibrosis, but as being at least in a majority of cases a dust fibrosis which, from its beginning as a clinically detectable condition, is linked up with an element of latent tuberculous infection. The after-history of these cases is in this respect significant. In the majority of instances silicosis is a progressive disease. The outlook of the individual case is mainly dependent upon whether the infective element remains "bottled up" and inactive, or whether, on the other hand, it becomes active, or a further infection occurs from outside the lung. In some instances advance is rapid; in others, advance may occur after a latent period of several or many years; in others, again, the condition may remain stationary practically indefinitely. But, although there are wide individual differences, dependent no doubt in part upon constitutional, nutritional and environmental factors, yet the tendency of the majority of cases to advance is, on the average, singularly uniform, and the culmination of that advance, if it proceed to its terminal stage, is in a form of obvious active tuberculosis in the silicotic lung, although by no means all cases advance to that condition.

A great deal of discussion was aroused in South Africa by statements made in a paper[19] by Dr. J. S. Haldane during a visit he made recently to study mining conditions in that country. According to Dr. Haldane the precautions taken against dry dust have been extraordinarily successful in reducing the risk of silicosis but the saturation of the air to attain this result aggravates the heat problem and makes the deeper mining of low-grade ore impracticable. He thinks this problem can be solved by mining in drier air, and the silicosis risk avoided by the addition to the quartzite dust of another kind of dust with the property of rendering silica harmless. In support of this he calls attention to the non-production of silicosis in British coal mines which are rock-dusted to prevent coal-dust explosions, although the dust used for this purpose contains about 35 per cent quartz and nearly 60 per cent total silica. The South African Mining and Engineering Journal devoted a series of editorials[20] to the points brought out by Dr. Haldane and their application to local conditions. The Journal states that while Dr. Haldane is convinced of the beneficial counter-effects of a "healthy" dust, the

19 Haldane, J. S. The Avoidance of Silicosis with Dry Methods of Working. Presented before the Joint Meeting at Johannesburg of Sections G and I, British Association, Aug. 2, 1929.

20 S. A. Min. & Eng. Jr. vol. XL, Part 1, Nos. 1976, 1977, 1979, August, 1929.

responsible leaders and controllers of the industry evidently hesitate to take the plunge. In support of his idea, Dr. Haldane said that the very dangerous menace to those working in Britain with refractory material, which is about 98 per cent silica, had been overcome by the admixture of only 10 per cent of a "healthy" dust. The Journal quotes also the testimony of Mr. J. N. Wynne, a mining engineer, to the effect that many white officials had worked continuously in the Kolar gold fields in India for 20 years and over and yet no case of silicosis had been recorded. Referring to Dr. Haldane's suggestion of the admixture of 10 per cent "healthy" dust to overcome the deadly effects of ganister, Mr. Wynne said that he was doubtful whether the admixture of the schistose wall rock in the Kolar dust was anything like as much as this.

Referring to Dr. Haldane's suggested remedy, the Journal said that there was no doubt in anyone's mind as to the improvement in health that had been brought about by wet methods. Moreover, contrary to a previous suggestion of the Journal, it was not proved that a wet atmosphere is weakening to the lungs as far as mining is concerned. However, on the other hand, it calls attention to the fact that the use of water has definitely not overcome silicosis, but only reduced its incidence. It has not made a complete job of it, and in that respect water has been a disappointment. Countering dust by dust may prove to be more in line with nature's methods, may give the quietus to phthisis, which all would welcome, and make mining, even on the Rand, what Dr. Haldane says he has always conceived it to be, a calling subject to some danger, but a healthy calling.

In closing the general discussion, Dr. Haldane said all that he desired to see was a thorough-going investigation especially directed towards determining whether in certain mines, such as Mysore, the alleged non-existence of miners' phthisis is due to the admixture of the silicious dust with other dusts. There are also other mining areas in which such conditions are said to be present, and these also should be investigated. He said he did not want to expose a large number of people to any dangerous conditions without previously having made reasonably certain that no great danger is really involved, but he would point out that nature is performing the experiment in spite of us and that miners' phthisis cases still occur on the Witwatersrand mines, in spite of wet mining.

The seriousness of the situation to the operators from the standpoint of miners' phthisis compensation is indicated by statements made in the South African Mining and Engineering Journal for July 6, 1929, in regard to expenditures made for compensation. According to the Journal, the administration of the miners' phthisis law is vested in the Miners' Phthisis Board, to which is attached the Miners' Phthisis Medical Bureau, functioning in association with the South African Institute for Medical Research in Johannesburg. In the last published report of this board, for the period April 1, 1927, to March 31, 1928, there is to be found a full account of the administration, together with sundry statistics and accounts covering the twelve months referred to. From the inception of legislative enactments until the end of the period mentioned, according to this report, an aggregate amount of £9,967,229 had been paid by the existing and previous Boards, in monetary benefits to silicotics and tuberculotics. As may be seen from a comparative statement, some 14,356 original awards had, at the end of the period under review, been made by the Boards to and in respect to miners (including inspectors of mines), and thus the average of the award, per case, was £694.

United States. - In New York City[21] an examination was made of 208 men exposed to rock dust in subway or tunnel construction. Silicosis was found to be present in 118, or 57 per cent of the men examined; 23 per cent showed radiographic evidence of ante-primary silicosis, 19 per cent of first stage silicosis, 7 per cent of second stage silicosis, and 8 per cent of third stage silicosis. The highest incidence of silicosis occurred among blasters and the lowest incidence among excavators. Seventy-six men, or 36.5 per cent had been exposed less than 10 years, while 132, or 63.5 per cent, had been exposed 10 years or more. The shortest exposure in the entire group was six weeks; in the ante-primary group it was three years; in the first stage group it was one year; and in the second and third stage groups it was nine years. The longest exposure in the entire group was 46 years. In the ante-primary group it was 40 years; in the first stage group, 46 years, and in the second and third stage groups, 44 years. It is apparent that the silicosis found in this study was of rather slow development, probably due chiefly to intermittency of exposure since the dust studies indicate that the silica dust hazard of the work in general is pronounced.

An investigation, begun in 1926, and reported recently, was made by Hayhurst and his coworkers,[22] in one of the largest and deepest sandstone districts in the world, located in Ohio and worked for more than 50 years. The workmen are employed by two quarry companies which market grindstones, scythestones, curbing, flagging, breakwater and building stone, and also furnace sand.

Pulmonary pathology was found in 55.1 per cent, including 38.5 per cent with silicosis in various stages, of 913 men examined. The total tuberculosis, however, was but 1.9 per cent, also general disability was low. These findings check relatively closely with those of the tuberculosis statistics for the community, according to the investigators. The period for developing silicosis, namely 16.24 years, is more than twice that reported elsewhere. The rock has an average content of from 92.15 to 97 per cent crystalline silica (SiO_2). The investigators also state that of 260 of the men having silicosis only 13, or 5 per cent, had tuberculosis as compared with the 20 to 30 per cent reported as usually found present by silicosis studies throughout the world. No explanation has been found yet for this anomaly.

A few years ago, the United States Public Health Service began a series of intensive studies on the health of workers in dusty trades. The first study was reported in 1928[23], and dealt with the health of workers exposed to silica

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21 Smith, Adelaide R. Silicosis Among Rock Drillers and Excavators in New York City. The Jour. of Ind. Hyg., vol. 11, No. 3, March, 1929, 92-96.

22 Hayhurst, E. R., Kindel, D. J., Neiswander, B. E., and Barrett, C. D. Silicosis with low incidence of tuberculosis. Jour. of Ind. Hyg., vol. 11, No. 7, September, 1929, 228-244.

23 Thompson, L. R., Brundage, D. K., Russell, A. E., and Bloomfield, J. J., The Health of Workers in Dusty Trades. I. Health of Workers in a Portland Cement Plant. Public Health Bulletin No. 176, 1928, 138 pp.

dust in a cement plant. The report on the second study, dealing with the health of workers exposed to silica dust in the granite-cutting industry, has been published recently.[24]

The study was of such a character as to present a rather definite picture of what happens to men working for many years under a dust hazard of the extent described. The salient points may be summarized as follows:

(a) The long period of service before the liability to tuberculosis becomes manifest (generally 20 years or more).

(b) The sharp correlation between length of exposure to the dust and the prevalence of tuberculosis and also the death rate from this disease.

(c) The close relation between the extent of dust exposure and the health of the men.

(d) The universal occurrence of silicosis among the workers.

(e) The large proportion of workers finally succumbing to tuberculosis.

(f) The almost invariably fatal form of the disease within a short time after the onset.

(g) The different character of silicosis as manifested by X-rays compared with that shown where there is exposure to a dust with a much higher content of free silica.

(h) The location of the tuberculosis lesion, usually basal, where the disease complicates silicosis.

(i) The absence of deaths from silicosis per se, tuberculosis apparently always intervening.

(j) The failure of workers to recover from their condition upon going into non-dusty trades.

(k) The high incidence of sickness of a severe nature from causes other than tuberculosis.

(l) The rising sickness and mortality rates from tuberculosis due to longer use of the hand-pneumatic tool.

(m) The high death rates at the present time from tuberculosis, compared with normal industrial experience.

During the fiscal year 1928-1929, the United States Bureau of Mines, in cooperation with the Metropolitan Life Insurance Company and the Tri-State Zinc and Lead Ore Producers Association, examined 9,208 miners in the Tri-State district of Oklahoma, Kansas, and Missouri, making a total of about 20,000 men examined since the beginning of this work in May, 1927. The tabulation of the histories of the men examined has been completed up to and including June 30, 1928.

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24 Russell, A. E., Britten, R. H., Thompson, L. R., and Bloomfield, J. J. Health of Workers in Dusty Trades. II. Exposure to Siliceous Dust (Granite Industry). Public Health Bulletin No. 187, 1929, 206 pp.

SUMMARY OF RECENT INVESTIGATIONS ON ABNORMAL AIR CONDITIONS

Abnormal Temperatures and Humidities

England. - On July 10, 1929, in a paper read before the Institution of Mining Engineers, Dr. Haldane gave a general review of the work of the Committee on the Control of Atmospheric Conditions in Hot and Deep Mines since its appointment in 1918. Sixteen reports to or by the Committee have been presented since that date. Dr. Haldane's paper, which is a summary of the investigations detailed in the 16 reports, was printed by the Colliery Guardian for July 12, 1929. According to Dr. Haldane, the first report contains a review of the existing knowledge of the conditions that cause physiological trouble from heat, owing to the rise of body temperature.

The second report deals with the question of the effect on the wet bulb temperature within a mine of the moisture content of the outside air. A description is given in this report of the refrigeration plant at the Morro Velho gold mine in Brazil used to cool the downcast air and at the same time reduce the moisture content.

The third report describes a new form of hygrometer developed for obtaining quickly and conveniently wet-bulb readings in mines.

The fourth and seventh reports give data on the temperatures of the strata and rate of increase of temperature with depth in various parts of Great Britain.

The fifth report contains an account of the heat and spontaneous fires met with in pyrites mines in Spain. This is a common condition in badly ventilated metalliferous mines in Great Britain but is much more striking where pyrites itself is being mined. In Spain in hot places, where the air was dry, the men were found working at temperatures of from 122 to 130°; the wet-bulb temperature or wet-kata readings were not available.

The sixth and ninth reports contain very important data as to the effects of heat on miners and their working capacity.

The tenth report gives confirmatory evidence as to the influence of seasonal variation in the moisture content of the outside air on the underground wet-bulb temperature.

The eleventh report describes experiments to define the relations between the wet and dry-bulb temperatures and the rate of air movement.

The twelfth and thirteenth reports describe experiments with local means to increase the cooling power in particular workings where for any reason there is difficulty in obtaining sufficient general ventilation.

The fourteenth report discusses available methods of measuring low air velocities which cannot be measured by an ordinary anemometer, and the important cooling effect of such velocities.

The fifteenth report contains a number of observations on the dry-bulb and wet-bulb temperatures and wet-kata readings in Indian mines. It appears that wet-bulb temperatures of over 85° F. with very little air movement are common in Indian coal mines. The limit for continuous productive work seems to be well below a wet-kata reading of 5.7.

The sixteenth report contains data on the reaction of unacclimatized persons to high temperatures and humidities. Such persons may be unable to produce more than 1 lb. of sweat per hour, so that with a good air current above body temperature their skin readily becomes dry and the body temperature rises.

The question of salt depletion by sweating was discussed at the 1929 summer meeting of the Institution of Mining Engineers, held at Nottingham, England. Mr. Hancock, one of Dr. Haldane's collaborators, reported[25] that experiments were carried out concerning the nature of the loss of chloride by the skin under various conditions, and the proportion of sodium to potassium salts. It appeared to the investigators that a sharp distinction must be drawn between two kinds of loss of water from the skin - namely, the evaporation of water that had permeated through the general surface of the skin, and the evaporation or trickling down of sweat secreted by the sweat glands. In the first kind of loss the skin apparently behaved as a semi-permeable membrane - permeable slowly by water, but practically impermeable by inorganic salts, such as the sodium chloride or bicarbonate of the blood plasma. In the second kind of loss - namely, by secretion of sweat - the glands produced a secretion of more or less specific composition, strongly hypotonic in comparison with blood plasma, but removing possibly large amounts of sodium chloride from the body. The first kind of loss depended largely upon the factors which determined the rate of evaporation from the unmoistened skin, and probably also upon the blood supply to the skin; the second kind of loss depended entirely on the activity of the sweat glands. According to Hancock, no other salts appeared to be present except in traces, although urea had been found in very definitely higher concentration than in the blood plasma. The sweat glands seemed incapable, beyond a very limited extent, of producing sweat which, like urine, varied in composition according as water and other constituents were superabundant in blood plasma or the reverse; their function was to regulate loss of heat from the body or communication of heat to it, and only to a minor degree did they regulate the composition of the blood. As regards prevention of symptoms due to excessive removal of chloride by sweating, all that was needed was a sufficient supply of sodium chloride. By eating red herrings or highly salted bacon, coal miners usually prevented the occurrence of these symptoms, and if stokers in hot climates at sea would add about one-tenth of sea water to the water which they drank during work - just as the coal miner added about one-fourth per cent to his water or tea - or would eat plenty of salt meat or fish, they could also prevent the symptoms. In some discussion on the report a Birmingham mining engineer stated that during long shifts in the pit at high temperatures, when

25 The British Medical Journal, September 7, 1929, p. 469.

...ling with pit fires, he had found that abstinence from water produced a curious irritation of the skin, and on drinking copiously he withstood the conditions much better. On the other hand, the president of the Institution, Professor Henry Louis, said that in the tropics he had made it a rule never to drink except at meal times, and when surveying in the jungles of the Malay Peninsula, where the conditions were both hot and moist, he had kept quite well, whereas a companion, who drank freely whenever he felt inclined, collapsed within three days and had to return.

France. - According to observations made by Boulin[26], in general workers exposed to hot and humid air become accustomed to it, if the temperature does not exceed a certain limit, and they do not suffer any ill effects. Involuntarily they eat less heat-forming food and accommodate their clothing to conditions while at work. Their movements are also slower and their coefficient of production lower than that of workers under other conditions.

Boulin calls attention to another phase of this problem, not usually considered, and that is that other phenomena, aside from inconvenience from the heat, may occur when men are working before incandescent materials, metals, or other bodies in fusion or at red or white heat. There may be not only elevation of the internal temperature or exaggerated transpiration, but modification of the cellular life or even destruction of the cells. According to Boulin, the incandescent hearth, a source of high temperature, gives off at times something else besides heat. He says that the radiations from a furnace heated to about 1,000° C. are caused by the accumulation of energy produced in a body raised to a high temperature and calls attention to the various kinds of rays that may be emitted, such as the Hertzian waves of wireless telegraphy, calorific infra-red rays, luminous rays of the visible solar spectrum, ultra-violet rays, the X-rays of Holweck and Roentgen, and the gamma rays of radio-active bodies. All of these radiations are propagated in a vacuum with a rapidity resembling those of light: 300,000 kilometers to the second. They reflect, refract, and polarize according to the same law. They all carry a certain energy which is translated differently according to the nature of the body that absorbs them. When radiations of a certain nature are absorbed and arrested, the energy that they possess is manifested in a special manner. Thus, the infra-red rays absorbed raise the temperature of the body that receives them, the luminous rays affect the retina of the eye, the ultra-violet rays exercise a chemical-physiological action; and the X-rays and the gamma rays have a destructive action in general.

According to Boulin, under certain conditions the radiations emanating from the open doors of industrial furnaces are abiotic, that is, they destroy or injure the cells. Most of these radiations are infra-red but, if the temperature is increased, the rays are luminous, then ultra-violet. At 400° C. the luminosity is very feeble, at 800° it is accentuated, at 1200° it is at "white" heat. At 1000° biotic as well as abiotic ultra-violet rays are produced, the number of abiotic increasing up to 1600°. Therefore, beginning with 1000° C., Boulin

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26 Boulin, P. La temperature dans les locaux de travail et dans certains chantiers. Ann. d'Hyg., N.S. No. 4, April, 1929, 213-230.

considers that anyone in the proximity of an important incandescent material may receive radiations capable of exercising an injurious effect upon the cells of the skin or the exposed parts of the body. This may even occur at 900°. This explains the sun-stroke and conjunctivitis of which men complain when working near furnaces or at electric or autogenous welding. This also explains the relatively increased number of glass workers affected by cataract. Boulin quotes Dr. Legge as saying that 17 per cent of glass workers examined by him had cataract in various stages, but he did not think the heat sufficient to account entirely for the condition, and Boulin mentions in this connection the sensation, to which Marboutin called attention, of burning resulting from exposure to feeble long duration ultra-violet waves, which may be a possible cause.

Boulin also calls attention to the similarity of heat and cold in their effects on workers, that is, both cause a diminution of production since a person with limbs numb with cold works involuntarily with less facility in the same manner as voluntarily he slackens his pace when humidity renders the heat difficult to bear. He says that, while there is heat stroke, which may be fatal, there is also "congestion from cold" which may have the same result. This latter is not of so frequent occurrence, however, as the ill effects of exposure to sudden changes in temperature, which causes pulmonary diseases, especially pneumonia. Examination of cases of premature disability shows a surprisingly large number attributable to temperature - 20 per cent of the cases of disability in Sweden are due to disease of articulation, and in Great Britain one-sixth of the cases of permanent disability among workmen is attributed to rheumatic affections, according to Boulin.

South Africa. - In discussing problems of ventilation in South Africa, Dr. Haldane[27] said that, while an acclimated miner is able to make far more use of his sweat glands and skin, in regulating his temperature, than an unacclimated man, there is a limit to what acclimation can do. As the amount of work done per unit of time increases, so, of course, must the minimum wet katathermometer reading at which body temperature, even with full acclimation, can be kept within normal limits; and here voluntary adaptation comes into play. A rise in mouth temperature beyond 100° F. makes a man not only uncomfortable, but also averse to, and unfit for, work. A sensible miner therefore avoids the work, or any conditions in which he cannot keep his temperature down.

According to Dr. Haldane, the great trouble met with from heat in the deep Rand mines, in spite of good ventilation, is the saturation of the air that is necessary to prevent risk of fire and shrinking of existing timber as well as silicosis among the miners, and if the saturated air could be avoided without increasing the risk of silicosis the temperature troubles could be greatly diminished and, thus, the much deeper mining of low-grade ore be made a practicable proposition.

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27 Problems Connected with Mine Ventilation and the Prevention of Silicosis. Jour. Chem., Met., and Min. Soc. of South Africa, Vol. 30, No. 2, August, 1929, 50-74.

Dr. Pirow[28], during the course of the discussion of Dr. Haldane's talk mentioned above, called attention to the complications in getting the workers acclimated in South African mines due to the fact that the native labor, which represents nine-tenths of the underground complement, serves for a very short period and then has to be replaced by fresh contingents of natives. In any one month 10,000 or more new recruits may be dealt with and the whole native labor complement is changed within a period of some eighteen months. In addition to these rapid changes in the native labor complement, the European labor changes amount to 5 or 6 per cent of the total complement per month. In spite, therefore, of the precautions that are being taken to acclimatize workers before they are put to serve in hot and humid atmospheres, Dr. Pirow stated that cases of heat stroke are by no means unheard of on the Witwatersrand. He emphasized that the mines on which cases of heat stroke have occurred or are likely to occur have enforced the system for acclimatizing the workers and are continually taking steps to improve the underground conditions. Nevertheless 17 cases of heat stroke have been recorded for the year ending June 30, 1929, of which 11 cases proved fatal. The matter, according to Dr. Pirow, is receiving the closest attention and further measures to prevent heat stroke are under consideration.

United States. - The subject of air conditioning indoors by mechanical means seems to be growing in interest, especially with reference to its influence on health and efficiency. In the past[29], air conditioning has achieved its prominent position largely because its application to manufacturing processes has carried with it some convincing dollars and cents arguments on why it should be used. Arguments of that sort are not so convincing when the result at stake is human comfort and health. The majority of people are concerned about their health only when they are about to lose it. The statement is made[30] that with engineers able to produce any desired conditions, doctors should be able to tell what the conditions should be and should corroborate the research promoted by the engineers to determine what they should be.

Bloom, in an article on cooling and humidifying buildings[31], calls attention to the tendency to maintain too low temperatures, and offers as a basis for indoor temperature and humidity for the summer schedule the principle that the necessary readjustment of the body heat controls in passing from outdoors to indoors and vice versa should be a minimum commensurate with suitable comfort contrasts. In support of this he mentions that a condition of 85° F. and 40 per cent relative humidity would be unbearable in a room during the heating season, and yet be gratefully comforting when it is 95° F. and 50 per cent relative humidity outside. The provision of reasonable contrast between indoor and outdoor conditions should be the objective. Excessive cooling in summer is just as objectionable as overheating in winter. If it is assumed that an effective temperature of 66° in the environment of an average person at rest and normally clothed is the condition wherein the body heat generated is thrown off with the greatest facility and the

28 Same reference as item 27, pp. 69-71.
29 Comfort and Health. Editorial in Heating-Piping and Air Conditioning, October, 1929, page 527.
30 Same as reference 29.
31 Bloom, S. C. Cooling and Humidifying of Buildings, Power, March 5, 1929, pp. 413-414.

least discomfort, then it would seem reasonable that the maintenance of an effective temperature indoors, midway between that prevailing outdoors and 66° would afford the maximum comfort contrast with the minimum disturbance in the body temperature control mechanism of those passing indoors and outdoors.

In a paper[32] dealing with the theory and practical application of air conditioning for the purposes of industry and to assure conditions of human comfort, especially in factories and auditoriums, Carrier classifies the subjects studied by various research organizations under three headings:

1. The relationships of the three variables, temperature, humidity, and air motion, to human comfort and the quantitative relationship of these variables one to the other.

2. The determination of zones of optimum comfort embodying these three variables in winter and in summer.

3. Determination of the relation of sensible and latent heat given off by the human body for various combinations of temperature, humidity, and air motion so that this quantitative data required for the design of ventilating equipment for human comfort in audience halls and theaters might be available.

For human comfort, Carrier considers that the range of satisfactory and practicable humidities is between 35 and 65 per cent, and the effective temperature for the maximum amount of work, according to data secured by the cooperative researches of the American Society of Heating and Ventilating Engineers and the U. S. Bureau of Mines and the U. S. Public Health Service, is between 40 and 75°. He also describes some of the many devices available for conditioning the air.

The following is a summary of the results of a cooperative study made by the American Society of Heating and Ventilating Engineers and the U. S. Bureau of Mines on therman exchanges between the human body and its atmospheric environment[33]:

A series of 267 tests was made on seven normal male subjects at effective temperatures of from 6.7 to 37.8° C., humidities of 20, 45, 70, and 95 per cent and in still air and air velocities of 1.19 and 1.96 meters per second (235 and 385 ft. per minute). The subjects had previously partaken of their regular diet, were normally clothed and seated a great part of the time, but were not restrained from normal movements of limbs and body. A minimum of 53.4 calories per square meter per hour was found for the temperature range of 19.3 to 28.7° C., a value in close agreement with results of other investigators for men under like conditions of activity and diet. For higher and lower temperatures there is an increase in the metabolic rate.

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32 Carrier, W. H. The Control of Humidity and Temperature as Applied to Manufacturing processes and Human Comfort. Heating, Piping and Air Conditioning, Vol. 1, No. 7, November, 1929, pp. 535-543, 605-616.

33 Houghten, F. C., Teague, W. W., Miller, W. E., and Yant, W. P. Thermal Exchanges Between the Human Body and Its Atmospheric Environment, American Journal of Physiology, Vol. 88, No. 3, April, 1929.

The effective temperature index of sense of warmth and physiological reactions for varying temperatures, humidities, and air motion is defined and heat production within and total heat loss from the body are shown to be functions of this index for both still and moving air.

Total heat loss from the body is divided into loss by evaporation and combined loss by radiation and convection, both of which are shown to be functions of dry-bulb temperature; the former increases and the latter decreases as the dry-bulb temperature scale is ascended.

Change from insensible to sensible perspiration takes place at 22.8 and 23.9° C. effective temperature with 95 and 20 per cent relative humidity respectively. The change in degree of perspiration with temperature and humidity is shown.

The human body maintains temperature equilibrium in atmospheres of varying temperature and humidity by controlling heat production at low atmospheric temperatures and by controlling evaporation by control of available perspiration at high temperatures.

According to Yaglou and Dokoff[34], the cooling laws of the katathermometer have been studied by many workers, but the range and combinations of the air conditions covered have been limited to temperatures below 100° F., it being generally assumed that the formulas applying in temperatures below 95° F. - i.e., when the kata cools from 100° to 95° F. - would hold also for temperatures exceeding 100° F. - i.e., when the kata warms from 95° to 100° F. The results of the investigation are summarized as follows:

The katathermometer has been calibrated for air velocities from 0 to 660 feet per minute, for dry bulb temperatures from 35° to 165° F., and for wet bulb temperatures from 32° to 112° F.

The dry kata is affected by dry bulb temperature and air movement, but for any given air velocity its cooling or its warming rate is not a straight line function of temperature, owing to varying influence of natural convection currents and varying effect of air density. Humidity has little or no effect on the dry kata so long as the dew point remains below 95° F. When the dew point exceeds 95° F. the dry kata cannot be used as an anemometer because water vapor condenses on its bulb. The usual changes in barometric pressure have no appreciable effect on the dry kata.

The wet kata is chiefly affected by wet bulb temperature and air movement. To some extent it is also affected by humidity and dry bulb temperature, particularly in very hot and dry atmospheres. The relationship between wet kata cooling or warming power and total heat content of air is not linear, as heretofore assumed; an equation of a higher order would therefore be necessary to express the relation.

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34 Yaglou, C. P., and Dokoff, K. Calibration of the Kata-Thermometer over a Wide Range of Air Conditions. J. of Ind. Hyg., XI, No. 8, Oct., 1929.

Owing to varying effects of convection currents and air density, it is very difficult to derive equations which would fit the experimental data accurately over a fairly wide range of air conditions.

For practical use, charts are given by means of which the air velocity can be computed from kata readings for any air condition within the experimental range.

SUMMARY OF RECENT INVESTIGATIONS ON SOME TOXIC GASES FOUND IN MINES

In studies made by the psychological laboratory of Mellon Institute of Industrial Research[35] it was observed that ordinary fatigue and the conditions of narcosis from drugs and from oxygen depletion depend alike upon asphyxiation of the muscle cells. An attempt was made to determine what degree of delayed response and impaired efficiency, as indicated by the physiological and mental signs, may by taken as an objective index of fatigue. Throughout all tests the powers of concentration, steadiness, persistence, substitution, calculation, memory, and the ability to sustain continuous reaction ran such close parallels with increasing symptoms of asphyxiation that, according to Mr. Johnson, head of the laboratory, no distinction existed between so-called "nervous" fatigue and physical exhaustion. Almost the entire range of disturbances observed related to factors in the physical environment, that is to say, to oxygen supply, air conditions, and comfort. A man gets drunk on too low an oxygen intake, asphyxiated by the combustion products within his muscle cells. According to Mr. Johnson, three conditions counterfeit this picture of alcoholic poisoning: (1) a diminished oxygen supply, due to great dilution of the air by means of other gases, or in rarefied air of high altitudes, or in inclosed spaces polluted by carbon monoxide; (2) a lowered rate of cell respiration caused by alcohol or other drug; and (3) induced fatigue, when cells burn faster than the toxic products of incomplete combustion can be washed away by the blood stream.

In England, during an inquiry into coal gas poisoning, a statement was made by the general register office that the number of deaths ascribed to coal gas poisoning in England and Wales had increased from a yearly average of 217 during 1911-1920 to 1,224 in 1928. The increase had occurred chiefly since the war and chiefly in suicides, the number of which had grown from 175 in 1918 to 1,086 in 1928. Accidental deaths had increased at the same time from 81 to 183. The statement was made that this form of suicide had only recently come into vogue. In 1901 there were only 7 such deaths but by 1910 the number had grown to 136 and in 1928 there were 1,086 deaths. However, attention was called to the fact that while during the last year there were nearly nine million individual gas supplies, there were only 130 deaths from accidental poisoning.[36]

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35 Drunk on Bad Air. Heating, Piping and Air Conditioning, November, 1929, page 616.

36 Deaths from Coal Gas Poisoning: The Popular Method of Suicide, Jour. Am. Med. Assoc., Vol. No. 93, No. 19, 1929, page 1483.

In a study on the comparison of the pathological effects of prolonged exposure to carbon monoxide with those produced by very low oxygen pressure, Campbell[37] found that the pathological effects of chronic carbon monoxide poisoning closely resemble those of prolonged exposure to very low oxygen pressure in the inspired air; the main changes he found were venous congestion with atrophy of certain cells (e.g. liver, heart muscle near epicardium), edema and dropsy indicating that heart failure is the cause of inability to tolerate prolonged exposure to carbon monoxide. The organs showing most marked congestion and its sequelae were the liver, lungs, heart muscle near epicardium, and the brain. Haldane's view, according to Campbell, that carbon monoxide acts purely by interfering with the oxygen supply to the tissues is supported. Power to tolerate carbon monoxide - just as in the case of low oxygen pressure in the air - depends upon the ability of the vital organs, particularly the heart, to continue to function under a low tissue oxygen tension; tissue oxygen tension is not brought back to normal level by so-called acclimatization under severe degrees of oxygen want. In another study,[38] Campbell found that during the first few days of carbon monoxide poisoning the tissue oxygen tensions are lowered to a more marked degree than by a similar loss of function of hemoglobin due to bleeding or hemolysis.

In connection with the possible use of mechanical refrigeration for cooling hot mines, a study was made by the U. S. Bureau of Mines[38] of the toxic effects on guinea pigs of air containing vapors of methyl bromide, methyl chloride, ethyl bromide, and ethyl chloride. In arranging the compounds in the order of increasing toxicity, ethyl chloride was found to be the least toxic and methyl bromide the most toxic. Methyl chloride and ethyl bromide occupied intermediate positions on the scale; in the respective order for short exposure to high concentrations; and in reverse order for long exposure to low concentrations. The gross pathology was quite similar for all four of these alkyl halides and was characterized by congestion, hemorrhage and edema of the lungs, and by injury to the vascular system as shown by the tendency toward hemorrhage. A distinct delayed effect of exposure was noted with methyl bromide and methyl chloride, to a lesser extent with ethyl bromide, but was not observed with ethyl chloride. The symptoms produced by methyl chloride, methyl bromide, and ethyl bromide were similar. For exposure to high concentrations of vapors, the symptoms were chiefly of an anesthetic character - that is, excitement, rapid loss of equilibrium, inability to walk, struggling and running motion of the legs. With low concentrations of vapors and long exposure the principal symptoms were weakness, rapid pulse, convulsive rapid respiration, with rales, and in some cases frothy (often blood tinged) exudate from the nostrils. The symptoms attending exposure to ethyl chloride were similar to the

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37 Campbell, J. A. Comparison of the Pathological Effects of Prolonged Exposure to Carbon Monoxide with Those Produced by Very Low Oxygen Pressure. The British Journal of Experimental Pathology, Vol. X, 1929, pp. 304-311.

38 Sayers, R. R., Yant, W. P., Thomas, B. G. H., and Berger, L. B. Physiological Response Attending Exposure to Vapors of Methyl Bromide, Methyl Chloride, Ethyl Bromide and Ethyl Chloride, Public Health Bulletin No. 185, March, 1929, 56 pp. The study was made in cooperation with U. S. Public Health Service, the Dow Chemical Co., and the National Research Council

foregoing, excepting that the signs of lung irritation were not pronounced. Excitement and restlessness appeared to be greater with the ethyl than with the methyl compounds. After exposure, the anesthetic effect produced by high concentrations disappeared rapidly, but symptoms produced by long exposure to low concentrations, such as weakness, rapid respiration and pulse, and rales increased and death usually ensued one to four days later.

Attention was called to the insufficient warning properties of the compounds and it was suggested that chemical warning agents be added to prevent serious voluntary exposure.

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DATA ON METAL-MINE VENTILATION IN 1929

BY

D. HARRINGTON

I. C. 6246.
February, 1930.

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

DATA ON METAL-MINE VENTILATION IN 1929[1]

By D. Harrington[2]

The year 1929 was chiefly one of progress as concerns the various factors affecting and being affected by ventilation in metal mines. As with all phases of health, safety, and efficiency in mining, however, not everything has been done which should be done, and by no means has perfection been reached in anything in connection with the ventilation of metal mines. One of the most noteworthy causes for congratulation is the fact that although a number of fires occurred in metal mines there was little or no loss of life in these affairs as far as can be learned. Numerous occurrences or conditions were brought to light during the year concerning factors entering into ventilation of metal mines, such as occurrence of gases in metal mines; effect of blasting on the air in mines; metal-mine fires, their occurrence and causes, and methods of prevention or of handling them; air conditioning; health of metal miners as affected by ventilation; up-to-date methods of forwarding and controlling air flow in metal mines; published data during the year relating to metal mining, etc.

GASES FOUND IN METAL MINES

As usual there were some ignitions of methane in metal mines or in tunnels; at least two ignitions of methane, fortunately with no fatalities occurred in the Hetch Hetchy Tunnels near San Francisco. In the 1928 report of the State Inspection Force of Kansas, published in the spring of 1929, appeared the following interesting description of a metal-mine explosion of methane with resultant loss of two lives:

> On July 16, 1928, a call came to the rescue department at Pittsburgh at about 3:50 p.m., advising that an explosion had occurred at a lead and zinc mine operated by the Cunningham Mining Co., located near Pleasanton, Kans. The explosion occurred about 2:30 p.m., and there were four men in the mine at that time, two of the number, namely, William Cunningham, owner, and Jack Howards were brought to the surface a few minutes after the explosion. The other two men were still in the mine.
>
> Mr. James Sherwood, State mine inspector, and George J. Gruber, clerk, proceeded to the mine in Mr. Sherwood's car. Mr. Hislop, superintendent of the central station, and Ernest Shaw,

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1 - Report of chairman of the subcommittee on Metal Mine Ventilation of the American Institute of Mining and Metallurgical Engineers, presented at the New York meeting on February 19, 1930. The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6246.

2 - Chief engineer, safety division, U. S. Bureau of Mines.

deputy mine inspector, and Mr. Glennon, deputy mine inspector, proceeded to the mine in the central rescue car. After driving as far as Arma, Kans., Mr. Glennon joined William Beveridge, superintendent of Arma station, in the Arma rescue car, and then the entire crew proceeded to the lead mine, which was a distance of 67 miles from Pittsburg. After driving two hours and five minutes we arrived at the mine, and were informed that Hugh Cunningham was hoisted up about 40 feet from the bottom of the mine, when he fell out of the bucket to the bottom of the mine. An employee of the mine went down and brought Mr. Cunningham to the surface. Mr. Cunningham received a fracture of the leg and forearm in his fall. He was immediately taken to a hospital, where he died a few hours later. Harold Cole was the only man in the mine when we arrived. At this time the gas was too dangerous to enter the mine, we were unable to descend with our apparatus on account of timbers and rubbish being strewn about the shaft, making it impassable. A canvas tube had been devised and after the fan had been run about an hour several men attempted to explore the mine. One of the number, Mr. Beveridge of the Arma rescue station, went to the bottom of the mine, and a few minutes later was again brought to the surface. He said that the mine was badly caving and pushing the cribbing and pump at the bottom of the mine to one side. He said it was unsafe to touch anything, as the mine was liable to cave any minute. Instructions were given to have all the rubbish cleaned out of the shaft, and attempts would then be made to find the body, as the mine was free of gas by this time. The essential thing to be done now was to clear the shaft of timber, etc., so that workmen could remove some of the rubbish at the bottom of the mine. This was partly accomplished and the water was pumped from the sump, and then a party of the rescue crew went to the bottom of the mine and found the body in the sump, where it had been lying under 4 feet of water. Harold Cole's body was brought to the surface at about 5 a.m., July 17, 1928.

This mine had not been operating very long when they broke into some old workings, from which the gas came. The miners used electric lamps, which were of the battery type. One of the men who was brought up alive said that he saw a rock fall and strike an electric lamp and then the explosion occurred. This mine was in a terrible condition and should have never been allowed to operate."

In an underground limestone mine in one of the Central States a ventilation survey disclosed that not only were there appreciable percentages of methane present in the air in which the work was being done, but there was also sufficient carbon monoxide to give the workers headaches and possibly under some conditions to produce more harmful effects; the methane probably comes from some carbonaceous shales which are present, and the carbon monoxide is due to the use of ordinary gasoline-operated trucks for transporting the limestone underground through some tunnels. While the occurrence of carbon monoxide underground from gasoline-operated trucks is novel, the occurrence of carbon monoxide in mine air from

gasoline-driven locomotives is not as novel as it should be, as a number of gasoline locomotives are operated in metal mines - a dangerous practice which should not be countenanced even when all available precautions are employed.

Every year brings forth some fatalities due to CO_2, or attendant oxygen deficiency, in metal mines, and 1929 was no exception; the following paragraph from the Engineering and Mining Journal, issue of April 6, 1929, describes a typical accident where due care is not exercised in going into abandoned mines or parts of mines:

> "At the Monarch-Rand property, in the Randsburg district of California, E. C. Marshall, superintendent of the mine, and J. A. Marshall, his brother, were fatally gassed on March 25, 1929. The property had been shut down for a considerable period, and the two brothers entered the mine, being lowered in the shaft. It is assumed that they were overcome by the gas and were unable to signal to the surface. A rescue party entered the shaft, but encountered bad air at the 600 level and was forced to return. After five hours the mine was sufficiently cleared to enable another party to reach the bodies of the Marshall brothers."

Since no year passes without the occurrence of fatalities when unsuspecting foolish people, and some times this includes mine officials, enter abandoned mine openings without exercising sufficient precautions, it would appear that not only abandoned mines but abandoned parts of operating mines should be kept effectively sealed. The following excerpt from The Black Diamond, August 10, 1929, indicates that this problem confronts not only metal but coal mines:

> "That old coal mines which have been abandoned should be sealed up, has been shown on several occasions in eastern Kentucky. Skeletons were found in an abandoned mine last year, indicating foul play, one being that of a girl about 16 to 17 years of age. Just recently two boys, 15 and 17 years of age, were found dead about 75 feet back in a deserted mine in the Hazard field, they having been overcome with bad air. Efforts of the district mine inspection department to revive the boys were without result. Both boys were high school students. They had gone to the mine with a horse and wagon for a load of coal, which was available for the taking. Even they might be crushed, through careless pulling of pillars. All of this surely bespeaks attention on the part of the authorities. An abandoned mine is a standing menace to a community, offering as it does infinite temptations to the criminal and infinite dangers to the careless. This matter is worthy of agitation until it is firmly fixed in the public mind."

A novel occurrence of CO_2 is described in The Colliery Guardian, issue of December 14, 1928, as follows:

> In a recent number of Montanistische Rundschau, an Austrian Mining Engineer, H. Drotz, gives an account of the serious disturbance to sinking operations caused by irruptions of carbon dioxide gas. The surface ground in the

> Ostrau-Karwin region (Moravia), where the two shafts - winding and ventilation - were sunk, consists of loam, sand and gravel, followed by the Eocene clay, which reaches down to the cap rock of the coal. In the lower sections and strata of the rock are interbedded Tertiary sand and grit. It was in the shifting sand at a lower level that the trouble was experienced with the irruptions of CO_2. After successive operations had been carried out extending over a period of eight years - from 1900 to 1908 - the shafts had finally to be abandoned and other ones sunk in the vicinity, away from the seat of the trouble. The sinking operations included ordinary sinking and, later, when the trouble became too serious to cope with, the Kind-Chaudron; while various methods were utilized for dealing with the water, including the use of compressed-air pumps and draining off the water by large-capacity buckets drawn to the surface by winding engines. The irruptions of CO_2 proceeded at intervals, frequently assuming a most violent form and causing serious damage to the then existing lining of the shaft, besides rendering the sinking and pumping operations difficult and dangerous owing to the risk of suffocation involved. Discussing the possible cause of the continued presence of the CO_2, the present writer ventures the opinion that it probably accumulated in a deep cavity in the rock and, owing to the superincumbent clay above, it could not find an outlet, so that it remained, at high pressure, in the cavity, which was exposed when this particular portion of the strata was uncovered by drilling operations."

During the year 1929, as well as in previous years, a gas carrying the odor of both H_2S and SO_2 occurred in a winze of a gold-silver property in the Rico-Colorado district. The analysis of one sample showed practically 50 per cent CO_2, about 10 per cent oxygen, and about 40 per cent nitrogen, the SO_2 and H_2S were not determined because of inability to analyze the gas immediately upon sampling and the fact that when analysis of samples with relatively small proportions of sulphur gases must be delayed, these gases break down. Here the gases were undoubtedly due to rapid oxidation of a granular pyrite, as there were locally high air temperatures (around or in excess of 100°F.) where the gas was at its densest.

In connection with some preliminary work being done in starting tunnels for the city of Detroit, two men lost their lives because of an occurrence of H_2S gas in a shallow shaft. During the past few years there have been a number of occurrences of methane, H_2S, CO_2, and other gases in the driving of tunnels for city water or similar service in various parts of the country, and it would appear that tunnel driving should be given more careful technical inspection and supervision.

During the year there has been a continuation of spontaneous fires in black slates in some Minnesota mines; the fumes have an odor of SO_2 and H_2S and also contain CO. Special methods of ventilation must be used in these places, and it is probable that ultimately an attempt must be made to put the region of the

fire and the adjacent territory under ventilating pressure in order, if possible, to force the gases into the strata more or less broken by mining.

In The National Safety News of September, 1928, there was an interesting article concerning H_2S and its treatment and effect as portrayed by a doctor who had treated more than 300 persons suffering from exposure to the gas. Below are salient paragraphs from this article:

Hydrogen sulphide gas attacks the mucous membranes of the body; the coverings of the eye, lining of the nose, throat, lungs, stomach, intestinal tract and urethra, all of which are protective. The mucous membrane may be either irritated or destroyed in varying degrees, regenerating, as a rule, within 48 to 72 hours.

The eye has nine coats, of which the mucous membrane is the outer, or protective, and the optic nerve with which we see is the inner. Hence the blurred vision when the mucous membrane is destroyed as light does not pass through normally, but no damage is done to the nerve layer. The danger in this state is infection from the outside, as from dirt, oil, etc., from the face of hands by rubbing. This infection might cause an ulcer to form and endanger the sight of the eye, as an ulcer from any other cause would do unless proper medical care is given.

Treatment for the Eye

Hydrogen sulphide gas, when it comes in contact with the eye, forms small amounts of sulphurous acid, which should be washed from the eye with boric acid solution, following with the neosilvol, 10 per cent and butyn solution 2 per cent every half hour or more often as the case needs, keeping either ice packs or hot towels over the patient's eyes and respiratory tract. Rest and sleep in a dark room is highly important. In very severe cases, atropine sulphate 1 per cent should be used and, if desired, small amounts of cocaine may be added to the solution of neosilvol butyn to ease the burning pain. The eyes should be covered and protected from the sunlight for 24 to 72 hours, as the case may require.

When the mucous membrane of the lungs is irritated or destroyed the patient should be protected from exposure to cold as pneumonia or other infections can easily gain a foothold while the mucous membrane or outside protective sentinel is being regenerated. The same is true of the stomach and intestinal tract. Nothing should be given by mouth except water as the stomach is or may very easily become irritated and cramping, nausea and vomiting instituted. In more severe cases, absolutely nothing should be given by mouth; even water will bring on relapses, cramping, and distress.

When a great concentration of gas is encountered, nature, ever on guard to save life, stops respiration; the respiratory center of the

brain is paralyzed, which stops the intake of the methane and higher gases into the lungs and, hence, into the bloodstream. The bloodstream itself is the part of the body affected by this heavy dose. If the patient runs to fresh air and respiration starts again of its own accord, or, if artificial respiration is instituted, the poison from the blood is eliminated through the lungs and kidneys, the latter evidenced by a burning sensation upon urination.

Resuscitation Saves Many

'Artificial respiration plays a big part in reviving men who have had a lethal dose of the gas. A worker was gassed last April and was "out" for two hours and fifteen minutes before any sign of breathing could be seen. He was kept warm, given a stimulant, and artificial respiration was admirably given by the men and nurse on the lease. Two hours and fifteen minutes is a long time to be dead physiologically but this man within four days returned to work. His color and general health gradually improved so that he was in good condition physically within two months. While resuscitation efforts in a few cases have been of no avail, the majority of such patients have been successfully revived within a few minutes to three hours.

It is inadvisable for one with an impaired sense of smell to work around the gas. For instance, one worker was gassed severely four times. Due to absolute blockage of the nasal passages, he was unable to smell the gas and walked blindly into it. He has had his nose operated upon and has not been gassed since.

Focal Infections Add Danger

It has further been observed by me, and by many other doctors with whom I am acquainted, that men with focal infections, such as pyorrhea, decayed and infected teeth, or very bad tonsils are much more susceptible to gas attack--eyes included--than men in good condition. Of course, this is only common sense and explains why we treat the same group of men nearly all the time, while others in the same gang seldom, if ever, are gassed. Therefore, it has been our conclusion that men with focal infections should, out of justice to themselves, have these cleared up before going to work where they might encounter high concentrations of gas. Any man on a tubing job or clean-out string is likely to have his eyes gassed and should, therefore, be in perfect physical condition at all times.

'Some are under the impression that tuberculosis, typhoid and other diseases are caused by gas. It is my opinion that gas might help kindle a latent or slumbering case of tuberculosis, as will cold, exposure, bad food, and poor nourishment, but it is not apt to do so. As there are no tuberculosis germs floating in the gas and one must necessarily have them in his system before tuberculosis can exist, gas could no more cause tuberculosis than it could cause gonorrhea, trench-mouth, or any other infectious diseases caused by bacteria.

FUMES FROM BLASTING

In connection with this matter of gases which may be encountered in metal mines, it should be borne in mind that practically every metal mine may at any time be confronted with essentially the same kind of deadly gases which caused the large number of fatalities in the Cleveland Clinic fire of May, 1929. The gases from the burning X-ray films were high concentrations of carbon monoxide mixed with heavy concentrations of various oxides of nitrogen, probably with heavy depletion of oxygen; these gases are almost sure to be found in the fumes of 20, 30, 40, 60 or 80 per cent dynamite such as is found stored in small or large quantities in more or less confined places in almost every metal mine and only too frequently surrounded with relatively few safeguards against fire or against unsafe distribution of fumes should a fire occur. It may be well to remember that concentrations of carbon monoxide or of dangerous oxides of nitrogen are very likely to be too great to allow of the safe use of any type of gas mask, and that in addition oxygen may be dangerously low; hence, nothing but the latest and most up-to-date types of oxygen breathing apparatus should be worn as a protection against breathing these fumes.

In a metal mine employing several hundred workers on a few levels in intensive development where relatively cool rock produced cool air in the working places and there was little or no dust, the matter of ventilation was largely ignored until conditions become almost intolerable because of smoke and fumes from explosives. The general manager finally admitted that there was a dead loss of at least 25 per cent of the efficiency of the workers (all on a day-pay basis). It was then decided to install a complete ventilation system with all of the fire prevention and other features involved, such as the concreting of at least one shaft and the sinking of another shaft. The installation of the system was to cost little less than one-half million dollars.

The excellent paper on ventilation at the United Verde mine presented at the New York meeting of the American Institute of Mining and Metallurgical Engineers in February 1929, brought out the fact that much of the urge toward the installation and maintenance of the exceptionally elaborate and effective ventilation system at the United Verde was the desire to safeguard workers from the deadly fumes met in the blasting of ore high in certain sulphides. The applicability of the ventilation system at the United Verde to the safeguarding of workers from blasting fumes is given in more detail in Mr. Glaeser's paper entitled "Handling Gas Hazards at the United Verde Mine" presented at the February, 1930, meeting of the American Institute of Mining and Metallurgical Engineers in New York.

Three factors in recent development of metal-mine blasting practice are of much aid in providing a safer and more healthful atmosphere for face workers. The use of electrical blasting is growing rapidly in at least some metal mining districts; electrical blasting automatically eliminates much of the noxious carbon monoxide, since the burning fuse has in the older system of blasting been responsible for much of the poisonous fumes. The more widespread use of clay or other similar material in tamping blasting charges also aids materially in keeping the air of blasting faces relatively clear of fumes. Finally, many mining companies have, for reasons involving safety as well as the health of the workers, required that much if not all blasting shall be done at the end of the shift; this automatically allows time for harmful dusts to settle and for explosives fumes to be drawn off by ventilating currents.

The Engineering and Mining Journal of March, 1929, contained the following paragraphs, indicating that one of the Ontario gold mines is using the South African "water blast" with intent apparently to remove the explosive fumes from faces after blasting; however, the water blast also has the very beneficial effect of not only settling out of the air the very fine dust due to blasting but also of converting settled dust on mine surfaces to mud, thus preventing this dust also from getting into the air:

"At the Wright-Hargreaves mine, in Kirkland Lake, it is the practice of the machine men to blow a mixture of air and water instead of air alone, after blasting, to expel the smoke and gas. To make this possible the water line is connected to the air line near the heading, so that water may be mixed with the air as desired. The mixture is blown through a 5/16 inch hole in a cap placed over the end of the 2 inch air line. This cap is screwed on by the driller after he has spit his fuses and is about to leave the heading. Connection of water line to air line is made in the following manner: At a point some distance back from the heading a tee is inserted in the air line. Into this tee at 90° to the line is screwed a nipple, on the end of which is an ordinary valve such as that used for regulating the water on the drilling machines. From this valve runs a 5 foot length of water hose, terminating in another valve, which is connected to the water line running to the face. These two valves are kept closed except when it is desired to blow the mixture of air and water.

The procedure followed by the machine man is as follows: When he finishes drilling, he turns off the air and water from his heading at the nearest junction point, after which he loads his holes and spits the fuses. Then he opens the two water valves in the connection with the air line just described (one of these is used as a check), leaves the heading, and turns on both air and water at the junction point. This arrangement requires that a pressure higher than that of the air be kept on the water line to force the water into the air line. This is accomplished by maintaining 125 pound pressure on the water line, as compared with 95 pound pressure on the air line. The difference is sufficient to permit the water line to be carried up into the stopes to blow the mixture of air and water there as well as in the headings.

Many of the metal mines of the United States could, with advantage to the health and efficiency of workers, adopt the previously described South African practice now apparently in use in some gold mines of Ontario.

METAL MINE FIRES

During 1929 there were the usual number of fires in metal mines or in tunnels, fortunately with little or no loss of life this year. These fires occurred in Michigan, Minnesota, Wisconsin, Montana, Utah, Tennessee, California, and other States. As usual the main, in fact almost the only, ignition agencies were open lights and electricity; yet there is only the most feeble of efforts apparent as yet looking toward safeguarding mines and tunnels and the workers in mines and tunnels from these dangerous agencies. In an excellent article entitled "More Light for Mine Workings" which appeared in the July 20, 1929, issue of the Engineering and Mining Journal, occur the following significant paragraphs giving at least qualified advocacy by a metal mining man of the use of miners' electric cap lamps in metal mines:

In contrast, the newer electric cap lamp is not well known as yet among the rank and file of metal miners. Just as the carbide lamp was slow in supplanting the candle and the sunshine lamp, its use at the start being confined largely to engineering crews and foremen, so time and better knowledge will evidently be required to put the electric cap lamp "across." For safety work the electric lamp is ideal. Where explosives are being handled, or stopes are hot, or the risk of fire is great, or where the air will not support combustion, it has decided advantages. It is more elaborate and expensive than its rival, and the weight of the battery as well as the lead and belt required are disadvantages save where a man's first thought is for his safety.

But it gives superior light. With further improvements particularly in lessening the weight of the battery, it will probably find place in metal mines at an increasing rate. What the ultimate features of this type of lamp will be, only the future can disclose."

In view of the far greater safety as well as greater efficiency of the up-to-date miners' electric cap lamps over any type of open lights yet used in mines, it is inexplicable that all of our metal miners and about half of our coal miners persist in exposing themselves to the really desperate hazards as to fires, explosions, and accidents (from insufficient light and air vitiation) which invariably are present underground when open lights are used.

In its efforts to prevent loss of life in mine fires, such as the one which occurred in the Hollinger mine in northern Ontario in 1928, Ontario has adopted some amendments to its mining law as indicated in the following items taken from the Canadian Mining Journal of March, 1929:

The Amendment to the Mining Act of Ontario which is the result of the Hollinger enquiry has incorporated the less stringent recommendations of Justice Godson into the Act, and it is understood that a further amendment will be brought down next year which will take care of other points. The additions for this year are quite simple, and do not inflict any hardship upon operators, and as they are the outcome of the deliberations, and advice of a number of the leading engineers in charge of mines in northern Ontario, they are probably already in effect in the majority of cases. The additions are as follows:

156a. Every man employed as an underground foreman (meaning thereby one who is exclusively engaged in supervising the work of other men) shall be able to give and receive orders in the English language.

156b. The Inspector of Mines shall have the right to suspend any foreman or mine captain who is not familiar with, or does not understand, the requirements of the regulations governing the operation of mines as contained in the Mining Act.

(11) No building for thawing explosives shall be maintained in connection with any mine except with the written permission of the Inspector of Mines. The site of this building and the style of structure

and equipment shall be subject to the approval of the inspector. The building shall be under the direction of the manager or some person authorized by him. The quantity of explosives brought into any thawing house at any one time shall not exceed the requirements of the mine for a period of 24 hours, plus the amount that it may be necessary to have thawing to maintain that supply.

(38a) All underground buildings or enclosure necessary for the housing and maintenance of machinery or equipment shall be constructed as far as practicable of fireproof material.

(38b) All fans except "Booster" fans shall be above ground and shall be reversible, and all fans and structures containing the same shall be fireproof.

(38c) Oil and grease kept underground shall be contained in suitable metal receptacles and the amount so kept shall not exceed the requirements for seven days.

(38d) There shall be a sufficient number of fire doors at every underground station where practicable so that the shaft can be completely cut off from the rest of the mine.

(38e) All inflammable refuse underground shall be removed at least once in every 24 hours, and shall be brought to the surface and there disposed of in a suitable manner.

(38f) Every shift boss and mine captain shall certify in writing to the mine manager at least once in every week, that there is no accumulation of inflammable refuse underground in the area under his supervision except as reported by him.

(38g) Suitable fire protection systems shall be installed at all underground crushers, tipples, and in dry shafts.

(38h) Every mine which has a maximum production of 100 tons of ore per day, shall be equipped with the necessary mechanical apparatus so that the stench ethyl mercaptan may be introduced into the air line as a warning signal to underground workmen.

(38i) Legible signs showing the way to emergency exits shall be posted in prominent places underground and all workmen shall be instructed as to location of auxiliary exits.

The foregoing regulations are excellent and constitute a considerable stride in advance of past practice, but it is unfortunate that they do not include a requirement covering suitably controlled mechanical ventilation.

One of the "lucky" fires of 1929 occurred in an underground explosives magazine. The fire was started either by an open light or by a cigarette or match used in lighting a cigarette, but fortunately the explosives burned instead of exploding. In connection with this affair attention is again called to the numerous hazards in the storage and use of explosives in mines; certainly open lights and smoking should not be allowed in or near any explosives magazine,

nor should persons using explosives smoke or carry an open light while handling explosives. Since many of the far too numerous accidents from explosives in metal mines originate in some manner or other from open lights or smoking, there is no good reason why both open lights and smoking should not be excluded from all mines. It may also be advisable here to call attention again to the fact that fumes from burning explosives, especially in a confined place such as an underground magazine, are likely to have so high a concentration of deadly gases, such as carbon monoxide and various oxides of nitrogen, as well as so great a depletion of oxygen, that none of the present-day gas masks are safe to use in or around the fumes. Hence, only up-to-date oxygen breathing apparatus should be used at these fires.

At a conference on spontaneous heating and ignition held in Washington, D. C., in November, 1929, attention was called to the fact that many substances such as sawdust, timber bark, splinters, etc., hay, straw, and other materials found or likely to be found in metal mines are subject to spontaneous combustion. It was also brought out that some of these substances in heating spontaneously give off explosive gases such as carbon monoxide, methane, and hydrogen, so that explosions are likely to accompany spontaneous heating, especially in a confined, poorly ventilated place. This fact is of importance in both coal and metal mining as indicating the necessity of "good housekeeping" in the removal of various kinds of rubbish from mine workings; certainly it is dangerous to store or otherwise dispose of various kinds of combustible materials in the "gob" as is so frequently done.

The American Mining Congress Committee on the Standardization of Fire Fighting Equipment in Metal Mines has been engaged in the completion of its report for submittal to the American Standards Association, and it is probable that these particular recommendations will soon be available as a "standard."

AIR CONDITIONING

Air conditioning as applied to metal mines has been largely confined to trying to cool the air where work must be done in high temperatures. To a limited extent removal of fine dust has been aimed at, but to a much greater extent the object has been removal of fumes from explosives. In the following data concerning court decisions affecting coal mines, there is a foreshadowing of what is very likely sooner or later to confront all classes of mines, coal and metal as well as nonmetallic; and it is interesting to note that in one instance the company was held responsible for alleged harm from chilling or cold conditions underground, while in another the blame was thrown on the company for a heat condition alleged to have affected the workmen. One item came from the Bulletin of the Anthracite Bureau of Information for June 1, 1929, and reads as follows:

> "The company objected to an award of compensation allowed by a referee. The claimant is the dependent widow of Joseph Gober who died in October, 1927, after he had worked in a wet part of a mine of the company.
>
> "The miner suffered a chill after standing in the water while at work driving a breast, and later died of pneumonia. The referee found death to be due to the conditions under which he worked and awarded compensation to the widow, and the company objected that the evidence

in the case does not prove that the deceased died as a result of accidental injury.

"On the question of whether exposure caused the fatal illness, the board found the evidence sustained the referee."

Below is a reading of the other case as taken from the Bulletin of the Anthracite Bureau of Information for July 6, 1929:

"The claimant is the widow of Frank Rucci, a miner who died last August, who was allowed compensation for herself and children. The husband on August 7, 1928, left home apparently in good health and reported for work as usual. He worked all day, although he complained of pain in his head. When he returned home he complained of illness. Less than a week later he died.

"The day that he was taken ill was extremely hot, and the board, in discussing the evidence, held that 'heat exhaustion coming in the course of employment has been held to be an accident within the meaning of the law.' The opinion of Commissioner Fleitz goes into detail relative to the medical testimony, and it stated in conclusion that the testimony of the physicians would seem to meet all the requirements as to evidential value required by the appellate courts and constitutes a preponderance of the credible testimony of record. The board affirmed the referee's findings of fact, conclusions of law and award of compensation.

An article in Coal Age of August, 1929, states that one of the anthracite mines of Pennsylvania maintains a plant to heat the intake air in winter in order to be able to operate its downcast haulage shaft to advantage. This is a decidedly commendable practice and is one which could be used to advantage by many of our coal and metal mines. There is no question that the main haulage shaft should always be in fresh air to insure safe escape of workers at time of mine fire, hence all haulage shafts by which men pass into and out of the mine should by all means be on intake air; this is difficult to achieve in winter in some regions, however, and the practice of heating the intake air as described in the article mentioned is, therefore, a good one. There is a possible alternative for the above practice, at least in some mines, in that where there are available shafts or other openings to the surface, the main intake and main return may be through some shafts or openings not used for man travel or at least for man hoisting; and the main haulage shaft may be placed on a small split of pure intake air taken from the downcast shaft at some point underground after the intake air has had an opportunity to become slightly heated by coming in contact with underground strata. This is a practice now being adopted by several large-capacity metal mines, which in some instances are providing additional openings to make the plan effective.

The problem of cooling mine air has not been particularly well solved as regards either individual working places or the entire mine. In some regions where there are high summer daytime temperatures and underground strata temperatures between 70 and 85° F., it has been found advantageous to operate the main fan of some metal mines during the hot part of the day just long enough to clear away the fumes from blasting. In some of these mines the surface air with summer daytime

temperature in excess of 90° F. increases the temperature of mine workings to such an extent as to make them decidedly uncomfortable, hence the main fan is operated only at blasting time and at night. Little or no progress has been made toward cooling individual working places, and the best method yet available undoubtedly is the supplying of adequate air movement or velocity at the point of actual work; the moving air had the natural cooling effect of evaporation induced by the passing or impinging air particles. In the DuPont Explosives Service Bulletin on auxiliary ventilation, issued in May, 1929, is a description of an inexpensive Venture blower which may be used to advantage in metal mines to provide a cooling air movement at or near the face.

In the foreign technical press there have been a number of references like the following, which was taken from the October, 1929, issue of the Mining Magazine of London:

> "Discussions have taken place on Dr. J. S. Haldane's proposals for avoiding silicosis by substituting dry mining methods for wet mining methods. Dr. Haldane holds that if the quartz dust could be mixed with shale dust the lungs would be enabled to work off the quartz dust which causes silicosis. He also stresses the fact that in deep mines water helps to raise the temperature, and when the temperature becomes too high miners become subject to heat strokes. Local opinion is that Dr. Haldane's scheme should be carefully investigated, but all experiments should be made under the strictest control. Mr. van Eyssen, consulting engineer to Messrs. Lewis and Marks, states that at the Lonely Mine, Rhodesia, many of the miners, both white and black, have worked there from the inception of the mine, yet not one of them is suffering from silicosis. This mine is a particularly dusty one.

To those who have spent any considerable amount of time studying the effect of dust in mine air on persons forced to breathe that air, it seems almost grotesque even to think of abandoning wet methods of mining, at least in so far as concerns the use of water while drilling, and in some mines while shoveling and before and after blasting. The attached item taken from the January 18, 1929, issue of Colliery Guardian, published in Great Britain, indicates that even coal-dust (held by most British and American dust authorities to be harmless when breathed by workers) causes not only illness but early death.

> A special verdict that Death was from pulmonary consumption caused by the inhalation of coal dust was returned at an inquest in London last week on George Albert Morgan (68), who was stated to have been a free miner, formerly owning a small mine at Brierley, in the Forest of Dean. The widow stated that her husband had been working underground in coal mines for 50 years. He was a free miner, having been granted the small mine because he had been born within two miles of Speech House, in the Forest of Dean. The mine, which he began working soon after the outbreak of war, had to be closed down about two years ago owing to failure of trade. Dr. R. M. Bronte, who made a post-mortem examination, said both lungs were jet black and very hard."

It would appear that if breathing air largely filled with coal dust (long held harmless and by some authorities to be helpful) brings about death, then it is manifestly foolhardy to advocate a mining policy in which decidedly large quantities of mineral dusts would be breathed by workers. Considered

strictly on the basis of efficiency much of the rock in our metal mines could not be drilled dry with even any fair degree of efficiency with any drills yet fabricated; moreover, it is very doubtful if many of our better-informed miners would entertain even the probability of working in the super-dusty atmospheres so familiar in the so-called dry mining methods of the past. The attitude of a large part of the metal-mining industry on dust in metal mines is reflected very nicely in the following editorial which appeared in the Engineering and Mining Journal of January 26, 1929:

"Dust removal is a matter of dollars and cents, from whatever point one views it. To install a dust-collecting system costs something much or little, depending on the specific application and the quality of the job. The amount can be determined easily by soliciting bids. Estimates will vary widely, ranging from that of the tinsmith who looks upon the task as a mere matter of constructing housings, with perhaps a fan or two at some carelessly determined point, to that of the engineer who thoroughly understands the problem involved, with its various ramifications. Here it should be borne in mind that, as in any other installation intended for permanent use, work of good quality will be cheaper in the end. Some dust-collecting "systems" have been worthless from the start.

"To do without a dust-collecting system will cost something also, wherever a plant has dusty air. This will be appreciated only when the operator comes to realize the ravages that dust can work. Perhaps it is because men have so long been looked upon as children of the dust, returning to it at the expiration of their days, that so many plant executives take dust as a matter of course. What is more natural than dust? We find it all about us. The wind blows it in our faces. The housewife is continually removing it from the furniture. So one should not be surprised if, in places, its presence in the air breathed by workmen from morning until night is ignored or considered lightly. Only when the condition becomes so obnoxious as to demand a remedy does it get attention.

"Wherein may dust, perhaps intrinsically worthless, cause a loss? That it must be injurious to health, if sufficient in quantity, every one knows. On this account alone it is safe to say that the day will come when in every State dust prevention will be prescribed by law as an ordinary precaution wherever men are engaged in work. The quicker industry is to adopt the remedy, the less onerous will be the legislation that must eventually be passed.

"Second, dust causes excessive labor turnover. No one prefers to work in dust and breathe it if another job can be found where the pay is as good and the conditions are pleasanter. Labor turnover and its effect on profits is a matter to which executives have given much attention. Dustiness in a plant will be found to be an influential factor. And even if dust does not actually drive a workman to seek employment elsewhere, it most certainly will make him less efficient by the annoyance and irritability it causes, in addition to its effect on his health in general. How illogical it is, moreover, in this day of the antifriction bearing and better lubrication practice - both devised to make machinery more efficient, to make it last longer, and

to consume less power - to permit a condition of dustiness to continue, when dust tends to prevent the attainment of these three objectives. Dust in bearings is responsible for a goodly portion of the annual repair bills.

In short, dust causes a lot of trouble about a plant, directly and indirectly, and its removal is worth considering. Many a company outside of the mining industry would not think of operating without the adjunct of a dust-removal system. That the mine operator, so efficient underground and in his metallurgical work, so often lags behind his brother executives in other industries in this respect is indeed curious.

The agitation in Europe and South Africa against so-called "wet-mining methods" is due chiefly to the fact that use of water at or near faces in mines with strata temperature in excess of 80° F. brings about a high relative humidity condition which in still air about 80° F. is oppressive and in temperatures above 90° F. becomes almost unendurable as far as performing efficient work is concerned. The remedy, however, certainly can not be the substitution of dry drilling or dry handling of material instead of wet methods, as there is absolutely no question that excessive air dustiness, whether in coal or in metal mines, ultimately means the death of those forced to breathe it. It would appear from the following that even in South Africa, where most of the agitation for abandonment of wet methods has taken place, some of those engaged in mining are relatively well satisfied with present methods of air conditioning in the mines, at least in so far as prevention of silicosis is concerned:

The Rand's gold mines are, in the central area especially, getting deeper and deeper, and there is no saying at present when or where the economic limit of depth will be reached. Already the West Sub. incline shaft of the Village Deep is fast approaching its final depth of 7,640 feet (1.4 mile) below the collar of the new Turf Shaft. The mean depth of stoping levels at the end of 1928 was 6,480 feet below shaft collar and the maximum depth of winzing below the 39th level was 7,360 feet. It will thus be seen that depths are fast being approached which will give rise to serious problems having no parallel on other mining fields to help in their solution. One of these problems is how the enormous pressures soon to be encountered are to be dealt with, which will probably mean the substitute of incline shafts for short verticals and cross cuts, shorter backs and narrow stopes worked on the panel system. The danger from rock bursts which, as it is, occur now and then and generally take their toll of life, will undoubtedly be increased and prove more formidable than the phthisis menace, which, though it has slain its thousands, has never yet closed down a mine, and thanks to the energetic and intensive policy of dust prevention in force is now well in hand, the dust content of mine air having been reduced to 0.8 milligram per cubic metre (the lowest average recorded so far) as against 5 milligrams in 1914.

The problem of providing suitable air for workers at faces where rock-strata temperatures are high, hence where the air is likely to be heated to or above the rock temperature and also to be highly humid because of naturally occurring water or water from wet drilling or other use, seems to be fairly well

met at the properties of one large metal-mining company and one coal-mining company by coursing relatively large quantities of fresh air from the surface or providing local methods of giving fairly high velocity to the air in the vicinity of the workers. The method used by the metal mining companies is described in an interesting article on Ventilation of the Butte Mines, Anaconda Copper Mining Co., in the Mining Congress Journal of October, 1929; the last paragraph of this article reads as follows:

> "For future ventilation requirements, consideration has been given both to the possibilities of further improvement of present measures and of artificially cooling the mine air, as has been proposed and, to some extent, practiced in other places. Technically, there are no unsurmountable obstacles to be overcome in applying such a method to the Butte mines, but its use depends more upon economic factors. These, of course, vary at different properties. In tropical or semitropical climates the surface atmosphere is likely to be of relatively little value in improving ventilation conditions within the mine, because of the initial high temperature and humidity of the air entering the mine. In the Butte district, surface temperatures and humidity are decidedly low and offer much greater opportunity for reducing the mine temperatures. Cost of artificial cooling will, of course, depend to some extent upon the process used, and will also vary widely with conditions at different properties. In general application to extensive workings in the case of any problem of deep mines can hardly be other than a matter of very considerable expense. Under exceptionally severe conditions, such as may occur in dead ends during the development of lower levels, the use of artificial cooling may be warranted. For the main ventilation system in the Butte mines it seems, however, that reliance upon present methods will economically be most advisable, at least for some time to come."

The conditions encountered and methods used in the coal mine are given in the conclusions of an article in the July, 1929, issue of the Journal of The Chemical, Metallurgical, and Mining Society of South Africa:

> "The author concludes as follows: Other Practical Measures.--- Of all the foregoing methods, the best results are obtained by doubling the air supply, since it is accompanied by a substantial increase in the air velocity, to which factor, as already mentioned, 67 per cent of the improvement in the KS reading is due. This influence of the velocity suggests the idea that, instead of doubling the volume of air, the velocity should be doubled by halving the cross sectional area of the airways, which would also reduce the transmission of heat from the rock by lessening the area of contact. The present practice is to make the galleries as large as possible, in order to reduce the frictional resistance to the flow of air; so that, in many cases, they could be reduced without impeding the haulage and without increasing the velocity beyond the permitted limit of 6 meters per sec. This practice has been adopted at the Sachsen Colliery for several years, with the result that the KS readings are maintained uniformly between 15 and 25, except occasionally at the face, where delays in stowing leave large empty spaces, and the air velocity is consequently low.

"Though the increased resistance lowers the capacity of the fans, this can be balanced by the smaller air losses through short-circuiting of the current. These losses are mainly due to leakage and the opening of the air doors, on the one hand, and to loose stowage on the other. Leaving the air doors open may cause the air temperature to fluctuate 6°C in a few minutes. Even new doors are not properly tight, and may leak to the extent of 10 per cent of the total air current, or even more when the current is large, in which case several doors should be arranged in succession. Other causes of short-circuiting are cavities in the goaf stowage and resulting fissures in the rock; in addition to which, the formation of CO_2 (by oxidation) in the goaf raises the temperature of the air passing therethrough. It has been found by repeated measurements that the total loss of ventilating air through short-circuiting may amount to as much as 25 per cent of the total, despite all preventive measures; and some pits report a loss of 60 per cent.

"The increased resistance due to lessened sectional area can be counteracted by smoothing the walls. For this purpose, some of the intake and return airways in the Sachsen pit have been lined with boards, thereby reducing the resistance in a considerable degree and increasing the volume of air supplied per minute by about 25 per cent, with the same consumption of power by the fan. Packing sawdust behind the boards reduces the transmission of heat from the rock, thus lowering the air temperature and improving the Katathermometer reading. The reduction in area of the intake airways can also be counteracted by dividing or widening the return ways; and where these latter form the sole haulage roads, the increased area lessens the resistance to the augmented volume of the return air (about 10 per cent above that of the intake, owing to the higher temperature and the accession of gas from the rock). At the same time the larger rock surface exposed and the formation of CO_2 from scattered coal results in a progressive rise in air temperature to the upcast; whilst, on the other hand, the temperature in the intake airways is reduced by about 4°C through the absence of CO_2 from scattered coal, and short-circuiting is lessened because doors can be set up anywhere without interfering with the haulage. The increased temperature of the return air improves the draught in the upcast shaft and therefore increases the ventilating current.

In opening up a deep pit, the development work should be carried on in a way calculated to enable the volume and velocity of the intake current to be increased, and the heat given out by the rock and oxidation kept down as much as possible by concentrating the work, reducing the number and length of the intake airways, and hauling the coal through the return ways. The air may be supplied to several districts from a main staple shaft, being split as near to the working places as possible, so that the rise in temperature is minimised. Development should be pursued in accordance with a time plan, with a view to minimising the heating effect of newly exposed rock. Places supplied with separate ventilation should be worked by a concentration of effort and only during the winter, the air brattices being insulated and the return taken to the main return ways by the shortest route; and in no case should two adjacent headings be driven at a time. Finally, development work should not be undertaken sooner than is necessary, since every needlessly exposed square yard of rock surface means increased air temperature.

The effect of rock temperature and heat of oxidation being greatest at the face, and the air velocity of the current therefore reduced, special attention must be devoted to minimum sectional area, short face length, and compact stowage. Not more than 30 men per shift should be employed in each ventilation section, in order to minimise splitting. Working toward home is advisable, in order to keep the air out of the goaf and to cool the standing coal; and loading direct into tubs is preferable to the use of face conveyors, the latter giving increased opportunity for oxidation and transmission of heat from the coal.

The foregoing measures have been largely adopted in the Sachsen pit, where they have resulted in improving the cooling action of the ventilating current and increased output; whilst the men who formerly worked in a naked condition and bathed in perspiration now wear clothing.-- F. Jansen, Colliery Guardian, Oct. 14, 1917. (J. C.).

That the problem of "conditioning" the air of mine working places is a complex one and that a suitable solution in one region may not necessarily be applicable to another or other regions is brought out by the following excerpt from the April, 1929, issue of Industrial Hygiene:

The observations on atmospheric conditions in coal mines in India recorded in this paper are of great interest to industrial hygiene. Generally the climate, especially in the hot season, is such in India that it is a relief to go below ground; and the coal face, the furtherest point from incoming air, is the coolest place. Nevertheless, conditions entirely unfavorable to white men are quite usual, and a wet bulb temperature of 85° F. is common, with the air practically still. In one place at 99° F. wet bulb, work had been going on regularly for months without obvious ill effect to the workers. Keeping in mind that 80°F wet bulb has been fixed as the upper limit for active work in Britain, such observations indicate that an Indian must be acclimatized to heat quite differently from Europeans. His low "minimal metabolism" and pigmented skin may be important factors. He dislikes a breeze and prefers 75°F. to 50°F. for work; and one group objected to leaving 88°F. with 98 per cent humidity for 20 degrees lower. He works irregularly; he may work steadily for an hour and then rest for fifteen minutes in a cooler place. Apparently effective temperature scales must be constructed afresh for such members of the human race. --E.L.C.

HEALTH IN AND AROUND METAL MINES

Adequate ventilation has long been known to be the most important step in any movement toward the improvement of the health of metal miners, and the following editorial taken from the Engineering and Mining Journal of Dec. 8, 1928, is a vigorous expression of the attitude of the leading metal mining journals on this subject:

The fine record of some mining companies in improved health conditions for underground and surface workers contrasts with the attitude of the majority - that it were best to ignore the subject rather than to invite attention to what might be judged as an unsatisfactory condition. Good ventilation underground is impracticable; the conditioning of air

is too costly; dust control is a fad; the spread of communicable diseases is inevitable - these are the arguments sometimes advanced in excuse for a continuation of conditions the effect of which is diminished output, failing morale, suffering, and loss. For hundreds of years the mining industries of the world have performed the ostrich act of burying the head in desert sand to avoid a candid recognition of what is obvious to the world at large. Underground mining is hazardous, but reports and statistics have proved that tremendous advances have been made in recent years in the reduction of accidents. This fine showing should be used as an inspiration and as an incentive for a mass attack on the problems of healthful conditions in mine and mill, by the provision of adequate lighting and ventilation, atmospheric temperature and moisture control, dust collection, medical examination, and hospital service.

"The old conception of requirements in the safeguarding of health was to have a company doctor, supplemented, perhaps, by a company hospital for curing the ills of employees and their families at little or no cost to the individuals principally concerned. Highly progressive companies sometimes maintained a dentist as well. More recently the safety engineer has come to be as important as the doctor. The prevention of accidents and the maintenance of good health are fundamental to efficient management and constitute a field in which the company doctor, the safety engineer, and the company officials, as well, should cooperate.

"The mining industries of the United States will do well to support the proposed National Institute of Health and encourage those researches on individual industry problems that have so potent an influence on the health of the country as a whole. Technical and economic progress in recent years has been due in large part to the commendable efforts to made to safeguard the worker; but much more needs to be done before the metal mining industry can be given a clean bill of health. The insurance of health and health insurance are not one and the same. Responsibility for the outcome of unhealthful conditions can not be evaded by the payment of premiums. Better to face conditions in their reality, meet the monetary loss, and determine to reduce the hazard, as has been done with accidents."

In connection with the numerous ills due or thought to be due to high-temperature air in mines, it appears that the boils which have been found to affect considerable numbers of miners (metal as well as coal) are now suspected of of being caused by high-temperature air or possibly high-temperature water. The following paragraphs taken from the July 19, 1929, issue of the Iron and Coal Trades Review bring up some conditions which should be investigated in the United States as well as in Great Britain:

"Mr. L. Holland said that, whilst he had nothing to say with regard to the physical side, on the practical side, whilst dealing with fires, he had noticed that some of the workers suffered from intense irritation of the skin. He wondered whether that was due to the loss of water or the loss of salt.

"Dr. Hancock said he had noticed that certain districts where sweating was common, irritation of the skin did occur, and he had met with instances of boils which were a source of irritation. He had also

noticed in those cases where boils occurred that the air conditions were not what they might be. The air was not as pure as it should be, and when one had heat, humidity, and poor air, there was a strong tendency to this incidence of boils. It was a question which needed investigation. As to whether the trouble was due to loss of water, or salt, or the air conditions, he could not say.

Boils have caused considerable trouble in some of the mines of Nevada and other states and usually the trouble has occurred in mines which are hot and wet.

The following data taken from an article in the Mining Congress Journal for November, 1929, give information concerning a health clinic which has been functioning with remarkable success in the Picher, Okla., mining region, where miner's consumption and other diseases of miners are decidedly prevalent. There should be many such clinics in our mining communities:

Along with the inauguration of the accident prevention work, a study of health conditions was begun by the United States Bureau of Mines. X-ray examinations revealed the widespread prevalence of silicosis and tuberculosis among the miners, many of whom had worked in the dusty mines of the Joplin District. The outcome of this study was a cooperative agreement reached in the early spring of 1927 between the Bureau of Mines, the Metropolitan Life Insurance Co., and the Tri-State Zinc and Lead Ore Producers Association, for the establishment of a well-equipped clinic at Picher, for the free examination of miners and other employees and their families, and also of applicants for employment.

Since the opening of the clinic, June 1, 1927, it has become increasingly evident that it is one of the most valuable assets to the work of accident prevention. It is obvious that illness and physical disability are apt to be fruitful causes of accident. In the first seven months of the operation of the clinic, 9,803 persons, including over 400 women and children, were given complete physical examinations, and the number being examined continues at about the same rate. The safety engineers have cooperated in every way possible to induce the men and their families to make use of the clinic, and the results have been very gratifying.

Following his examination, each man is given a clinic card containing his photograph, physical rating, and instruction as to when to report for reexamination. He is given proper advice, and, in certain cases, treatment. As a result, many men have been taught to take proper care of their health, many have left the mines for open-air employment, and others have been transferred to jobs more suitable to their physical condition.

It has not been the policy of companies to discharge a man on account of low physical rating, but rather to assist him to improve his condition. An applicant for employment, however, is asked by most companies to show his clinic card, and any one with a serious infirmity or a contagious disease is not employed. This precaution is a very valuable protection not only to the company but to the employees. Most of the miners have come to realize the value of the clinic to them and do not hesitate to make use of it.

CONCLUSION

Metal-mining people have never been more inclined toward systematic ventilation of their properties than now; this is manifested in numerous activities, not only of individual companies but also of organizations concerned with the improvement of conditions affecting ventilation of metal mines. One of these manifestations is the work of the American Mining Congress Committee on a Code for the Ventilation of Metal Mines, which is now engaged in "rounding out" the proposed code or standard.

Numerous inquiries and discussions concerning various details entering into the ventilation of metal mines have been made; some of these are; What is the correct type of main fan or of auxiliary fan? What is a good type of ventilation door, fire door, or ventilation regulator? What is the most effective lining of a main air-carrying shaft? Should the main fan be a blower or should it be used as an exhaust unit? Should the fan be placed so as to allow ready reversing of air direction? These and a large number of additional questions have been discussed, although a definite answer has not always been forthcoming. For instance, there is a wide difference of opinion as to the construction of a satisfactory type of fire door - one which will resist fire, will not warp or sag, will close tightly and leak little, will not decay or rust or otherwise disintegrate, and can be used from day to day if necessary.

In the Iron and Coal Trades Review, issue of March 22, 1929, a short article entitled "Life-Saving in Colliery Explosions and Fires," discusses some of the advantages of the use of doors and particularly of emergency doors on intake air courses to be operated at time of fires for the protection of employees who may be on the interior of the mine and on the return air side of the fire or explosion. There is absolutely no question that well-placed doors may be utilized to advantage in saving employees in both coal and metal mines at time of fires and probably to a lesser extent in explosions. The following paragraph taken from the article in the Iron and Coal Trades Review summarizes the points claimed by the principal proponent of the use of doors.

> (1) That the men inby will make use of the life-saving door when the need arises; (2) that when the life-saving door has been shut, and even if the fan is not stopped or 'slowed,' the air in the return will not become irrespirable by leakage of smoke into it, before the men at the face have had time to come out; (3) that facilities for reversal of the ventilation are not an effective safeguard; (4) that providing materials for erecting a barricade is likewise not an effective safeguard; (5) that the specified position--in the intake, just outby of the first branch road to the face--will rarely be in unsettled ground; (6) that it will be practicable to keep the door--in that position--reasonably air-tight, and that even a very leaky door will be far better than no door; (7) that the door would have been an effective safeguard to avert each of the ten worst disasters in Great Britain in recent years, which have been described in the Paper and discussions, and in which 990 men and boys were in grave peril, of whom 481 were killed by smoke from a fire in the intake.

THE GROWING ADOPTION OF MECHANICAL VENTILATION IN METAL MINES

Metal mines are adopting mechanical ventilation much more extensively than in the past, and this movement has naturally been accompanied by an increased use of such so-called coal-mine devices or adjuncts as overcasts, undercasts, regulators, splits, doors, brattices, and even curtains for air deflection. Thoughtful and far-seeing metal-mine operators are now providing and maintaining air courses not only for intake but also return or used air, even sinking shafts, driving raises, drifts, etc., wholly for ventilation. Sealing of abandoned workings is being done in some mines to conserve air supply or to prevent air vitiation.

Although many mines hold main haulage shafts on return air, there are numerous instances where the far safer system of holding main haulage on intake or at least fresh air is maintained. In general, present-day fan installations at metal mines allow reversing the direction of air currents, but unfortunately only a small percentage of the mines install doors in such manner as to be able to take advantage of this reversing feature if it is needed. Some of the wide-awake metal-mine operators are using smooth-lined, fireproofed main air-carrying shafts, thus reducing fan power consumption and at the same time providing the most effective type of fire insurance. Other mines are manipulating the air currents (where they are thoroughly under control- this means of fan, doors, overcasts, etc.) to prevent fungus growth, or even to kill fungus where it gets a start. In fact numerous metal-mining companies are now taking advantage of the almost limitless benefits obtainable in an up-to-date mine from a well-installed and well maintained ventilation plant.

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MINING METHODS, PRACTICES, AND COSTS OF THE CANANEA CONSOLIDATED COPPER CO., SONORA, MEXICO

BY

WILLIAM CATRON

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February, 1930

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS, PRACTICES, AND COSTS OF THE CANANEA CONSOLIDATED COPPER CO., SONORA, MEXICO [1]

By William Catron [2]

INTRODUCTION

This paper is one of a series being prepared by the United States Bureau of Mines on mining methods, practices, and costs in the various mining districts of the United States. The mining methods described are those used in the mines of the Cananea Consolidated Copper Co., a Mexican corporation whose holding company in the United States is the Greene Cananea Copper Co.

The mines are at Cananea, Sonora, Mexico, 25 miles directly south of the international boundary, approximately in 31° north latitude and 110 to 130° west longitude. Cananea is connected with the boundary towns of Naco and Nogales, Ariz., by a standard-gage railroad.

The population of the Cananea district is approximately 15,000. The mines employ about 1,300 men underground and 300 on the surface, while the concentrator and smelter employ about 900 more. The current production of the three producing mines, the Colorado, Capote, and Veta Grande, is about 2,500 tons of ore per day, working on two 8-hour shifts.

From the beginning of operations by the present company in 1901 to the end of 1928, a total of 20,059,229 wet tons of ore has been mined, with a metal production of 1,032,147,402 pounds of copper, 22,754,573 ounces of silver, and 144,308 ounces of gold.

ACKNOWLEDGMENTS

The author is indebted to T. Evans, general manager of the Cananea Consolidated Copper Co., for permission to write this paper. He likewise wishes to acknowledge the assistance of the company staff and of C. H. Johnson, assistant mining engineer of the United States Bureau of Mines.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6247."

2 - One of the consulting engineers, U. S. Bureau of Mines.

HISTORY [3]

Little is known of the early Spanish workings. Francisco Velasco, writing of Sonora about 1845, states that the mines were worked before the eighteenth century by the House of Guea, and later by Jose Perez of Arizpe, the capital of the district. Finally, it is said that in 1865 General Ignacio Pesquiera, with an escort of 500 soldiers, established a camp at Ronquillo, where the present smelter is, and erected a small smelting plant at the place now known as Cananea Vieja. Mining and smelting were carried on in a desultory way for about 15 years. About 1881 a Cleveland company built a smelter at the foot of the hills some 10 miles west of the mines at Cajoncito. Ore averaging 20 per cent copper and 15 ounces of silver was smelted and operations carried on for about two years. The smelter was then moved to Puerticitos, but was abandoned after another year.

The properties around Cobre Grande at the southeast end of the Cananea Mountains were bonded to Charles Benham of New York, up to 1886, and later to Major Morton in the interest of F. Augustus Heinze. After taking out about $110,000 in copper matte, this bond was given up because of the death of Major Morton.

Colonel W. C. Greene then secured control of practically all the mining properties in the Cananea district. Failing to obtain private capital to develop them, he organized the Cananea Consolidated Copper Co., Mexican corporation, and later floated the stock of the Greene Consolidated Copper Co., the holding company, on the New York stock market. This subsequently was merged with the Greene Cananea Copper Co.

The first smelting furnace was blown in May, 1899; the first concentrator was built in 1901, and the narrow-gage railroad connecting this plant with the several mines of the district was completed in 1902.

The great Capote ore body was struck in May, 1900, and the Oversight in December of the same year. The Veta Grande and Chivatera ore bodies, also lying within the Capote basin, were opened at later dates, as were a number of outside mines, such as the Cananea-Duluth, the American, Kirk, Cobre Grande, Sierra de Cobre, Elisa, Henrietta, and Puerticitos. In 1926 the Colorado ore body was discovered.

Figure 1 is a map of the district.

GEOLOGY

The rocks of the central part of the Cananea district are largely of igneous origin. The oldest rocks, however, are limestone and quartzite, probably of Paleozoic age, of considerable thickness but small areal extent. These are overlaid by thick, undifferentiated volcanics. Both sedimentaries and volcanics are intimately intruded by acid and basic igneous rocks, from which the ore deposits have been derived.

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3 - Emmons, S. F., Cananea Mining District, Sonora, Mexico. Economic Geology, vol. 5, No. 4, June, 1910.

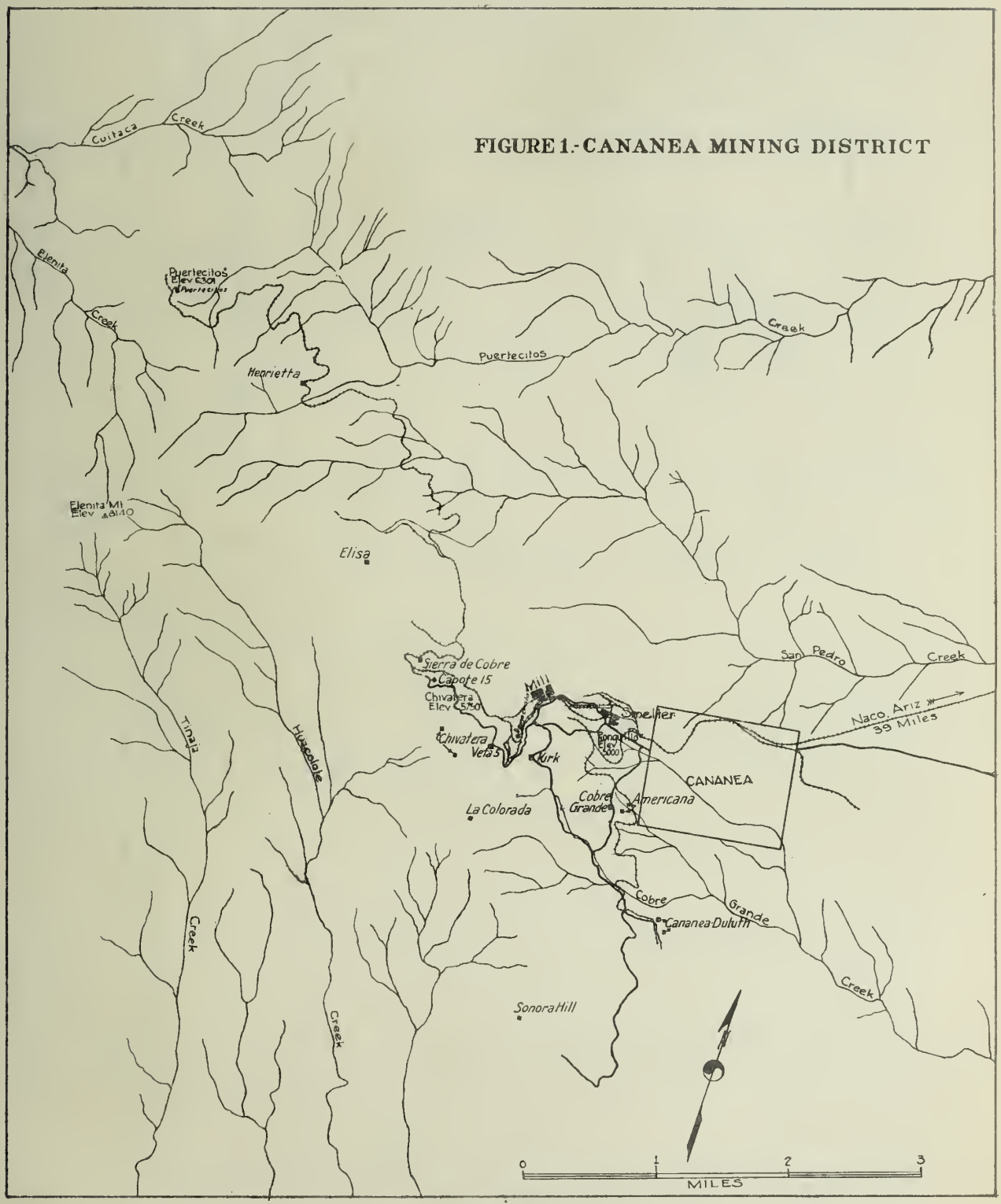


FIGURE 1.-CANANEA MINING DISTRICT

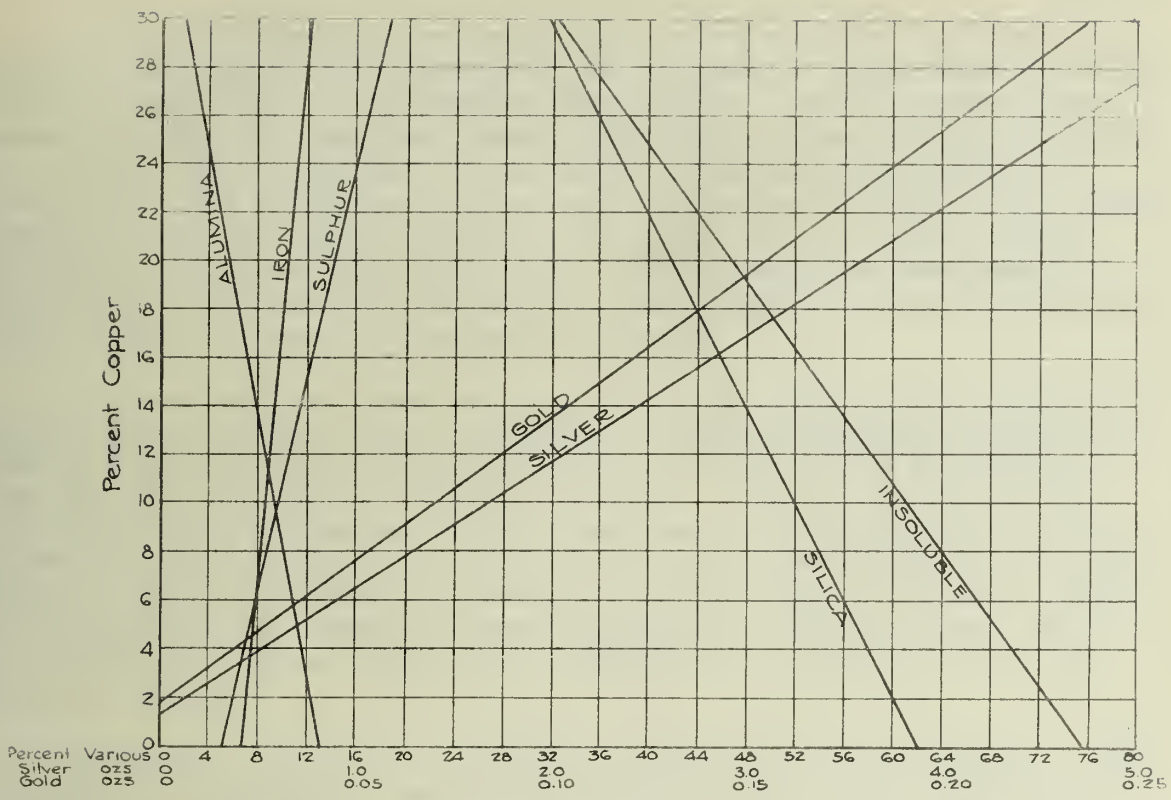


FIGURE 2. – VARIATION OF VALUES AND ANALYSIS OF COLORADA ORE WITH COPPER CONTENT (BASED ON TOTAL RECEIPTS TO DATE AUGUST, 1928

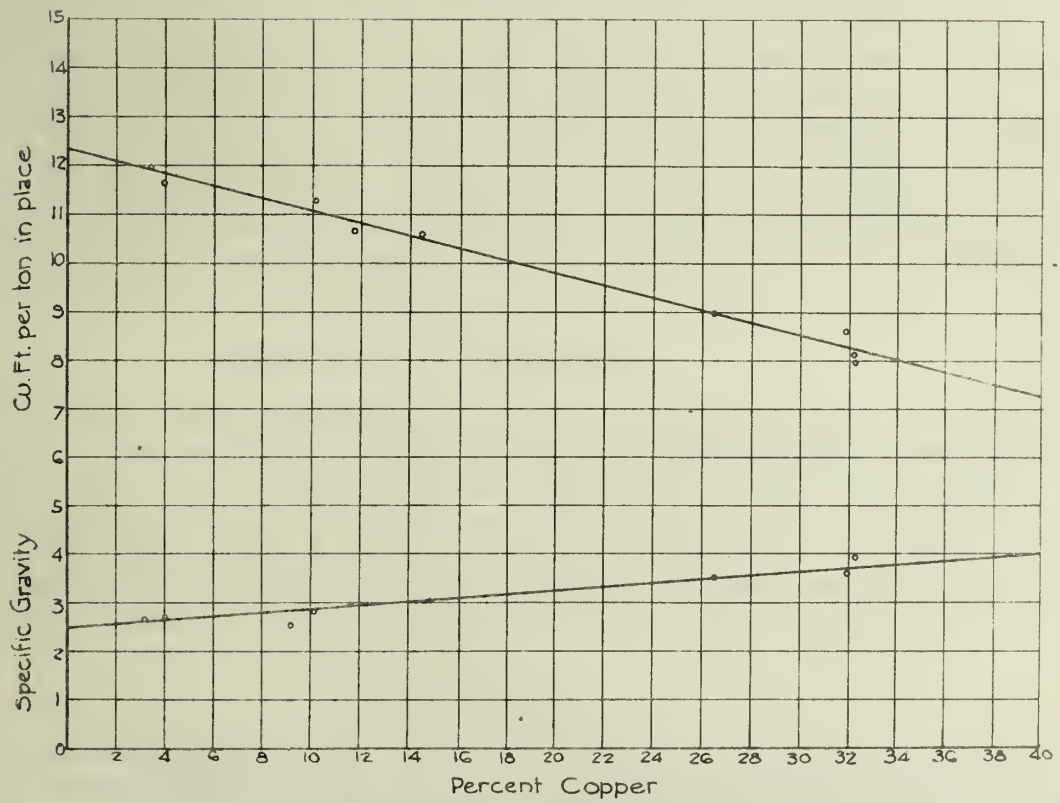


FIGURE 3. – LA COLORADA ORE. SPECIFIC GRAVITY AND CU. FT. PER TON IN PLACE. VARYING PERCENT OF COPPER

The ore bodies are of various types, often difficult to classify, ranging from distinctly high temperature to intermediate temperature deposits in limestone, quartzite, tuff, granite, and porphyry. Ores in garnetized limestone have been of great importance in the past, though none is now being mined; they were chiefly primary ores, though of unusually high grade, and were very spotty and irregular in occurrence. Occasionally shallow enrichment produced small but very rich deposits. The ore shoots were steep-dipping, with irregular and often indefinite walls. Stope boundaries were generally assay walls; the wall rock was not uniform, and though hard, was often seamed with slips and joints which made mining difficult.

The ores of the Capote and Veta Grande mines are of the intermediate temperature type. In this case secondary enrichment has played an important part in the formation of economic ore. The ore bodies are of irregular outline and have no well-defined structural control. They lie 50 to 500 feet below the surface in altered and highly-fractured ground which caves readily. The primary sulphide minerals are pyrite, with a small admixture of chalcopyrite and in places of considerable amounts of zinc blende. Secondary enrichment is responsible for large shoots and chimneys of chalcocite ore. With depth, particularly in the Capote mine, primary chalcocite and bornite are important ore minerals.

The deeper primary ores occur as vertical pipe-like deposits. The upper levels of the Colorado pipe are characterized by a ring of ore surrounding a waste core. This ore ring converges until on the bottom levels the waste core disappears and the pipe becomes a solid elliptical ore area. The mineralization follows well-defined zones of brecciation which have approximately circular or elliptical outlines. However, the stope boundaries are always determined by sampling, as the copper minerals extend outward from the main ore body as a network of small veinlets and make low-grade ore of the wall rock adjacent to the pipe. Careful stope sampling is therefore necessary. The ore minerals of these deposits are chalcocite, bornite, chalcopyrite, tennantite, and tetradherite. In the Colorada, molybdenite is abundant and at another similar deposit, the Cananea-Duluth, both sphalerite and galena are common. Quartz is the principal gangue mineral. Figure 2 shows the composition of varying grades of Colorada ore.

PROSPECTING AND EXPLORATION

Prospecting and exploration at first consisted of sinking shafts or driving tunnels on mineralized outcrops or in the vicinity of the prominent iron-stained gossans which were considered indications of secondary enrichments below. Later, and continuing to the present time, a great deal of drilling has been done in the district. Prospect drilling of new areas from the surface has been done by churn drills; diamond drilling has been carried on underground, generally in exploring or developing known ore.

Churn Drilling

A considerable amount of churn drilling was done along the southwesterly extension of the Ricketts fault in an attempt to find a lateral extension of the Oversight ore. The Colorada and Sonora Hill areas were churn-

drilled on coordinates in a systematic exploration campaign. Holes were drilled on the corners of 200-foot squares. At first, when interest was concentrated chiefly on possible secondary ores, holes were drilled to depths of only 300 to 500 feet. Later, upon discovering the existence of the rich primary ores, many holes were drilled to depths of 800 to 1,600 feet.

The holes did not need to be cased, except a few in soft ground, as in the Oversight area. The holes were sampled at 5-foot intervals, all the way from the surface. All the sludge was run through a splitter, saving a 1/4-part sample. This sample was dried, without settling or decanting, over a fire, and the sample assayed for total copper. Composite samples of 40 or 50 foot lengths were assayed for gold and silver. A geologist inspected the sludge and noted the kind and character of rock being drilled through.

Diamond Drilling

Diamond drilling has not been used in the past for primary surface prospecting. At the Elisa mine several holes were drilled to determine the geologic structure of an unprospected area. At the Capote vertical and inclined down holes were drilled from the bottom level to test the extent of the primary ore. In the Colorada mine a body of low-grade ore was developed by horizontal holes laid out on regular coordinates on two levels.

The bit used in diamond drilling gives a 1-1/2-inch hole and a 13/16-inch core. No casing has been used in the diamond-drill holes. Occasionally a hole is grouted to prevent loss of water. The sludge is run through a riffle-type sample-splitter which takes a one-eighth split. The sample is put in a compressed-air sample drier or filter, which removes most of the water. This filter was adopted after trying tubs and settling launders, which, it was thought, caused errors by losses during decanting or by concentration of the copper minerals in the baffles of the launders. The filter has the advantage of being sturdy and compact, and is easily and quickly operated. Since it may not hold all the sludge and water of the split sample from a given length of hole, it may be necessary to blow out the water two or three times to get a complete sample. Heavy canvas is used for the filtering element; the water comes through practically clear. The dried samples are weighed and assayed for total copper by the permanganate method.

The core is not removed at regular intervals, but is taken out whenever the core barrel is full or the rods are pulled for any other reason. Each section of core removed is handled as a unit. The percentage of the core recovered is noted each time any is removed from the hole. The core is carefully inspected every day by a geologist, and notes are taken for permanent record. A small piece of core is taken for a permanent sample and the remainder is sent to the assay office where it is ground, weighed, and assayed for total copper by the permanganate method. If gold and silver assays are desired, they may be run on composite samples of the entire hole or certain portions of it, or on individual cores. Chief reliance for knowledge of the value of the ground drilled is placed upon the core assays; the sludge serves as a check. If a combined value is desired, it may be calculated either according to relative weights of core and sludge or according to percentage recoveries and volumes.

Core recoveries vary greatly in the district. In the Colorada mine recoveries have ranged from 40 to 95 per cent, averaging 75 per cent; in the Elisa mine, in hard limestone, recoveries averaged more than that figure, while in the soft Capote ground they were generally under 50 per cent.

"Long-Hole" Drilling

A great many prospect holes have been drilled in the Elisa and Colorada mines with a heavy drifter-type machine. Its principal application has been the delimiting of ore bodies, serving the purpose of crosscuts through the veins, thus assisting in the laying out of development work. The holes have all been flat and have varied in length from 35 to 125 feet, averaging 85 feet.

The drill is similar to a drifter except that it has a swiveled water connection ahead of the drill chuck. A 1-1/2-inch shank back of the swivel fits the machine chuck; the front end of the swivel receives the threaded shank of the 1-1/4-inch drill steel. The shank of the swivel connection is retempered after each hole. Mine water is always available under sufficient head and no pump is used. The drill is set on a cross arm between two columns. One and one-quarter inch hollow-round steel is used in 6-foot lengths with a few 3-foot lengths for making changes. The steel is connected by sleeve joints. The bits are formed on 3-foot lengths of steel; the starter bit is 3 inches in diameter and the succeeding bits decrease in size by one-sixteenth inch. The sludge is all saved, settled in tubs, and the water decanted. The drill crew consists of two men. Twelve holes, totaling 1,180 feet, were drilled in 64 shifts, including the time required to move and set up, or an average of 18-1/2 feet per shift.

SAMPLING AND ESTIMATION

Drifts and crosscuts are pick-sampled at 5-foot intervals when in waste. When there is doubt whether it is waste or low-grade ore, channel samples are taken. These are cut horizontally, sometimes with a moil and hammer, but usually with a compressed-air chipping hammer. Grab or pick samples are taken from the face only for current information. Pick samples for permanent records are taken from the raises. Samples for control of stoping are taken by a crew which works on a special shift, overlapping both day and night shifts. Assay results from these samples are painted on the timber or the rock wall of the stope on the day following the taking of the sample. This prevents mining waste or leaving good ore, as it is often impossible, especially in the Colorada, to distinguish between ore and waste. When stopes are finished or before fill is run into a mined-out floor, another crew taken pick samples along the walls for permanent record on the assay maps.

Mined ore is sampled before being dumped into pockets or bins at the shaft. At the Capote, for example, a grab sample is taken from each car as it reaches the shaft station. At the Colorada, motor cars of ore are sampled before being dumped into the pockets at the shaft. The 1-ton cars hoisted in cages to the 450 level are sampled there. Small pieces of wood are marked with the stope numbers and placed on the cars before they are caged to show the

source of the ore. The car samples, as well as the pick and channel samples from stopes and development work, are sent each day to the sample mill at the Capote mine. There they are crushed, dried, ground, and split, and the resulting small samples are sent to the assay office at Ronquillo. All mine samples are assayed by the permanganate method. By means of the methods of sampling in use, the number of cars and the tonnage and grade produced by each stope and by the mine is known for each day. These figures check the returns from the mill and smelter sample plants closely enough for purposes of control of production.

Methods of estimating reserves vary according to the data available. In the caving blocks the ore outline is laid out on the assay map of the level or the undercut floor to include all ore possible that runs above shipping grade and exclude any that is under. All assays within the ore outline are then averaged to give an average grade; the area is computed or measured. Products of areas by grades on the level in question and the one above are then combined to give an average area and grade, and multiplied by the height to give cubic feet.

Information gathered from all previous stoping or development assays was used in making estimates of ore in blocks laid out for cut-and-fill or other methods. Either vertical or horizontal end areas may be calculated. For the initial stopes on a level, drifts and crosscuts are usually the only sources of information available. Allowance must be made for the position of the drifts, if the ore is in the shape of a vein with a high-grade core and low-grade walls. High assays may or may not be scaled down, depending upon the nature of the ore. Pillars can be estimated with considerable accuracy, as usually there are assays on both sidesand at every floor.

The conversion of cubic feet to tons was formerly calculated so as to allow for sorting. In the Capote, for example, 15 cubic feet per ton was at one time the factor used. Figure 3 is a chart which shows the relation between grade and cubic feet per ton of Colorada ore and which is used in making conversions from cubic feet to tons.

METHODS OF DEVELOPMENT AND MINING

Development

General Development. - Because of the hilly nature of the country, the mines of this district usually have been developed by both shafts and tunnels. The Veta Grande is worked through a vertical shaft and a tunnel, and has underground connection with other mines. The Capote is at present operated through a 5-compartment vertical shaft, with underground connections to the Elisa and other mines. A former operating shaft and adits on the upper levels of the Capote are now abandoned, or serve only for ventilation. The Colorada is opened by a vertical shaft from the surface, with two adits on the 500 level (see fig. 4). On this level ore is hauled from the shaft to the railroad ore bins, waste for fill and mine timber and supplies are taken in, and the men travel from change house to shaft. Main tunnels or adits are generally driven 8 by 8 feet in the clear, and drifts in which tramming is

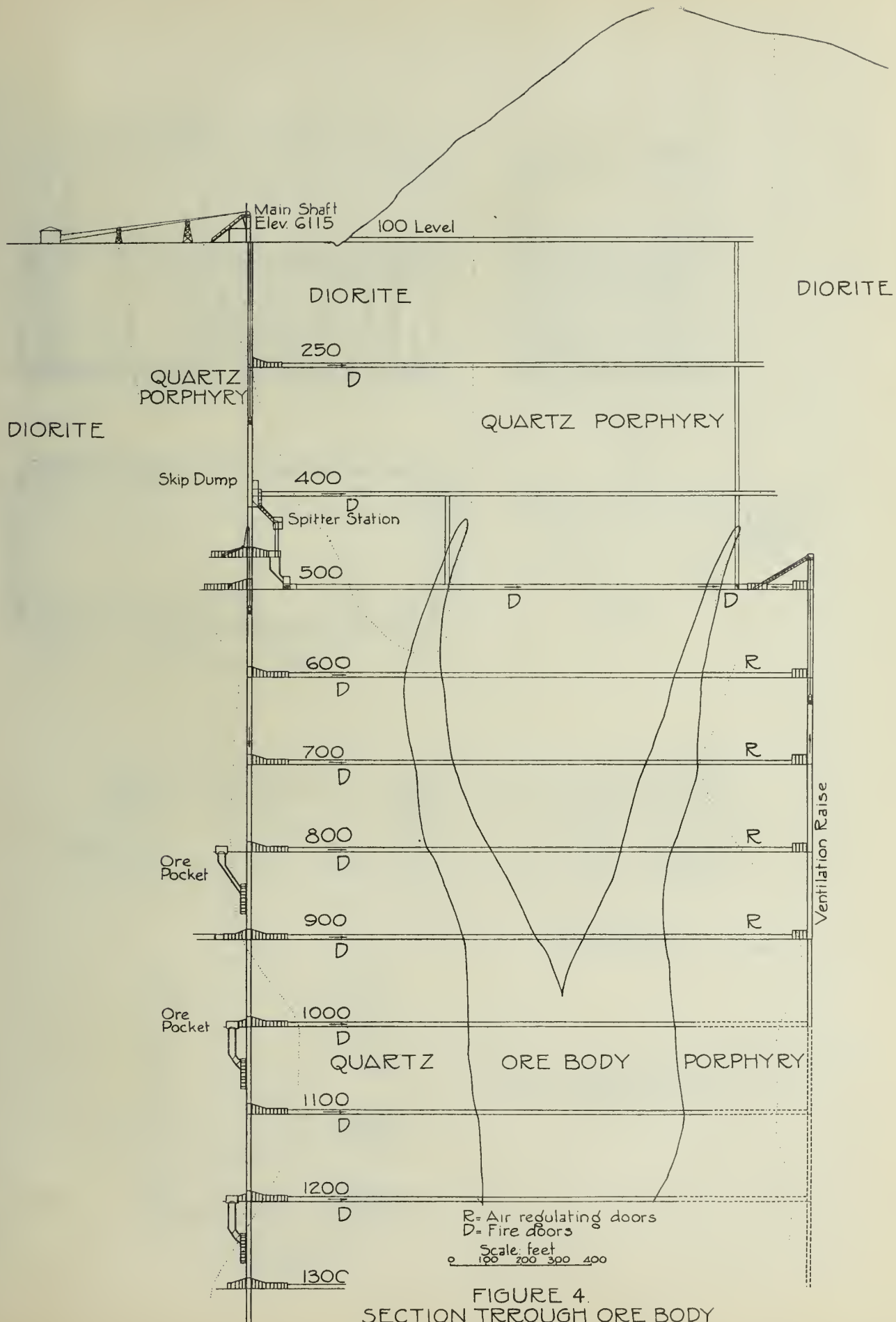
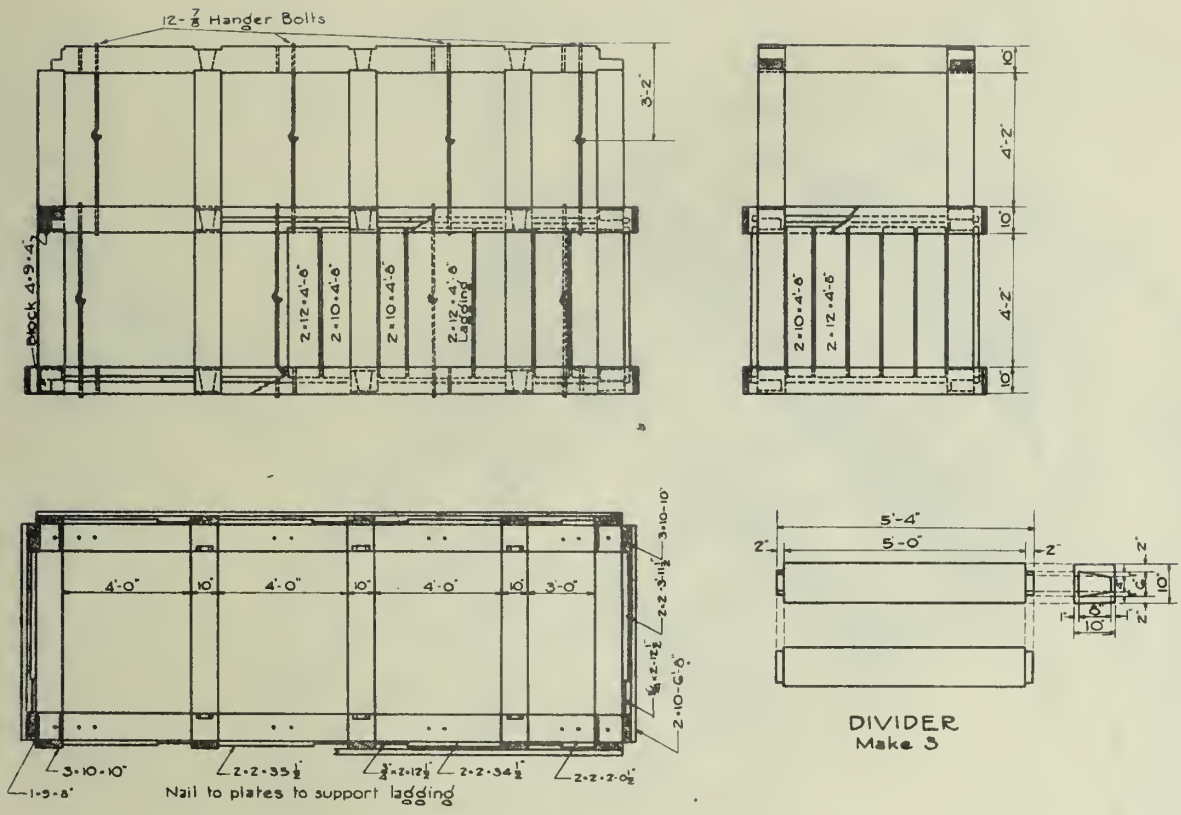
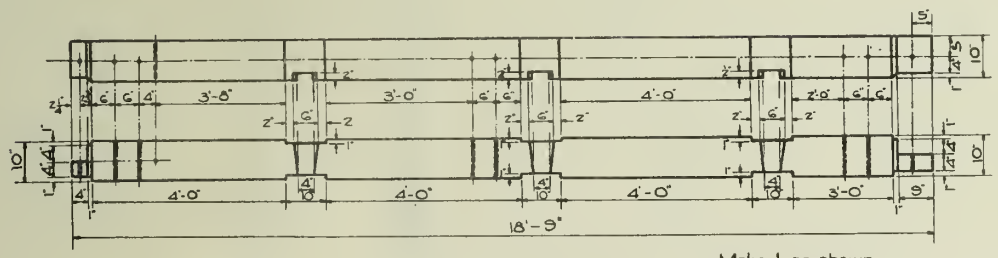


FIGURE 4.
SECTION TRROUGH ORE BODY
"LA COLORADA" MINE
CANANEA, SONORA, MEXICO





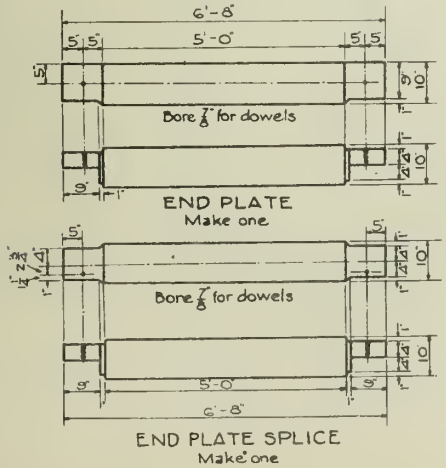
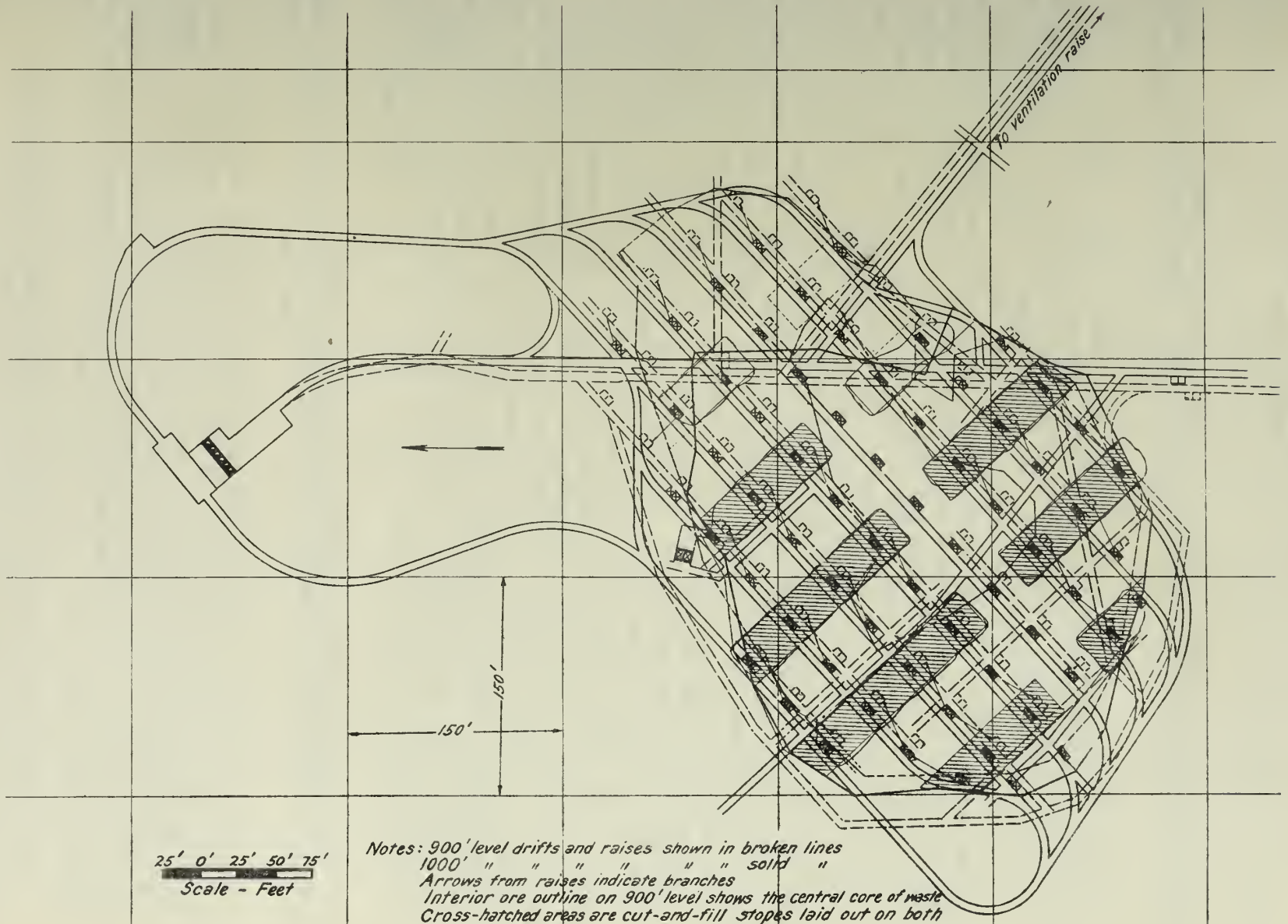


FIGURE 5. - 3 1/2 COMPARTMENT SHAFT TIMBERING PROVIDING FOR ENLARGEMENT



Notes: 900' level drifts and raises shown in broken lines
1000' " " " " " " solid "
Arrows from raises indicate branches
Interior ore outline on 900' level shows the central core of waste
Cross-hatched areas are cut-and-fill stopes laid out on both levels.

Figure 6.- Typical haulage and development levels

done by hand are 6 by 8 feet. All shafts, with the exception of the Veta, which has 4 by 4-1/2 foot compartments, have hoisting compartments 4 by 5 feet in the clear.

The opening of the Colorada ore body for production required about two years of intensive work. The following account of this work will illustrate the general methods of development:

In July, 1926, a drift was started from the Kirk mine, on what is now the 500 level of the Colorada. A month later the Colorada shaft was started. From the surface to the 300 level the shaft was three-compartment, timbered with 8 by 8 inch timber. Below this point 10 by 10 inch timber was used. At the 450 level a fourth compartment was added at the manway end. On this level a small hoist room was cut out and a hoist installed to operate a sinking cage in the third compartment. The counterbalance occupied part of the new fourth or manway compartment. A pocket for handling rock brought up in cars on the cages was built between the 500 and 450 levels. Below the 500 level the sinking of the shaft was paralleled by the sinking of a two-compartment winze in the ore. As soon as possible, connections were made between the two on each level, to provide ventilation and additional entrance for men and supplies. Below the 500 level, the levels were spaced uniformly 125 feet apart. At each level the shaft crew cut a station about 20 feet back from the shaft before carrying the sinking further. This allowed drifting to be started when desired without interrupting work in the shaft. During the two-year period, the four-compartment shaft was carried to the 1,100 level, and two skip compartments were added from the surface to the 900 level. The latter was done by raising from each level to the one above, several such lifts being worked at the same time. When sinking, wall plates framed with half-length tenons on one end were used; the spaces left by the short tenons were filled with blocks. Then, when raising and timbering the skip compartments it was easy to remove the blocks and fit the new wall plates into position. Figure 5 shows the timbering of the initial three and one-half compartments. At the time the skip compartments were being driven, the present steel headframe was built, replacing the wooden sinking headframe without interruption to operations. Upon completing the shaft, a pocket was cut on the 800 level. This was done by driving a small inclined raise from the shaft to the level, afterward enlarging the raise to full pocket size.

As rapidly as possible each level was explored by drifting, and haulage drifts were laid out according to the information thus gained. Figure 6 shows a typical haulage level.

On the 500 level the main drifts were driven 8 feet high and 8 or 9 feet wide. Work was carried on in three shifts per day; drilling was done over the broken rock pile from horizontal bars. Trolley locomotives were used for tramming. At first 2-ton cars on 18-inch track were used; afterward the gage was changed to 36 inches on this level, and 7-ton bottom-dump cars were used for ore and 5-ton rocker-dump cars for hauling waste fill. At that time, some of the drifts were widened. The main adit is now 8 by 10 feet in most places.

Below the 500 level some of the main drifts from the shaft to the ore body were likewise driven with all possible speed. Drilling was done over

the muck pile, and four muckers employed at each face. Only 1-ton cars and hand-tramming have been used for development work on these levels. Sometimes the drifts were double-tracked as advanced, to give a good supply of empty cars at the face. On the 800 level, which is the haulage level for all stopes between it and the 500 level, the 1-ton cars have been replaced by 4-ton gable bottom cars and electric haulage has been installed. Track gage, however, has been kept at 18 inches. Most of the drifting, even through the ore, was done without timbering. When drifts are timbered, as in places under the stopes, square-set timber with a 6-foot cap is used, making the drift about 5 by 8 feet in the clear. This is large enough for either the 1 or the 4 ton cars.

Raises were driven from level to level in accordance with stoping plans. These raises were of three kinds: Square-set, cribbed, and stulled (figs. 7 and 8).

On the side of the ore body opposite to the Colorada shaft a large 3-compartment shaft was put up from the 800 to the 500 level. This was intended primarily for ventilation and an emergency exit. One compartment is a manway, however, and cages have been installed in the other two, with a hoist on the 500 level. The shaft thus serves for an emergency exit and for handling men and supplies.

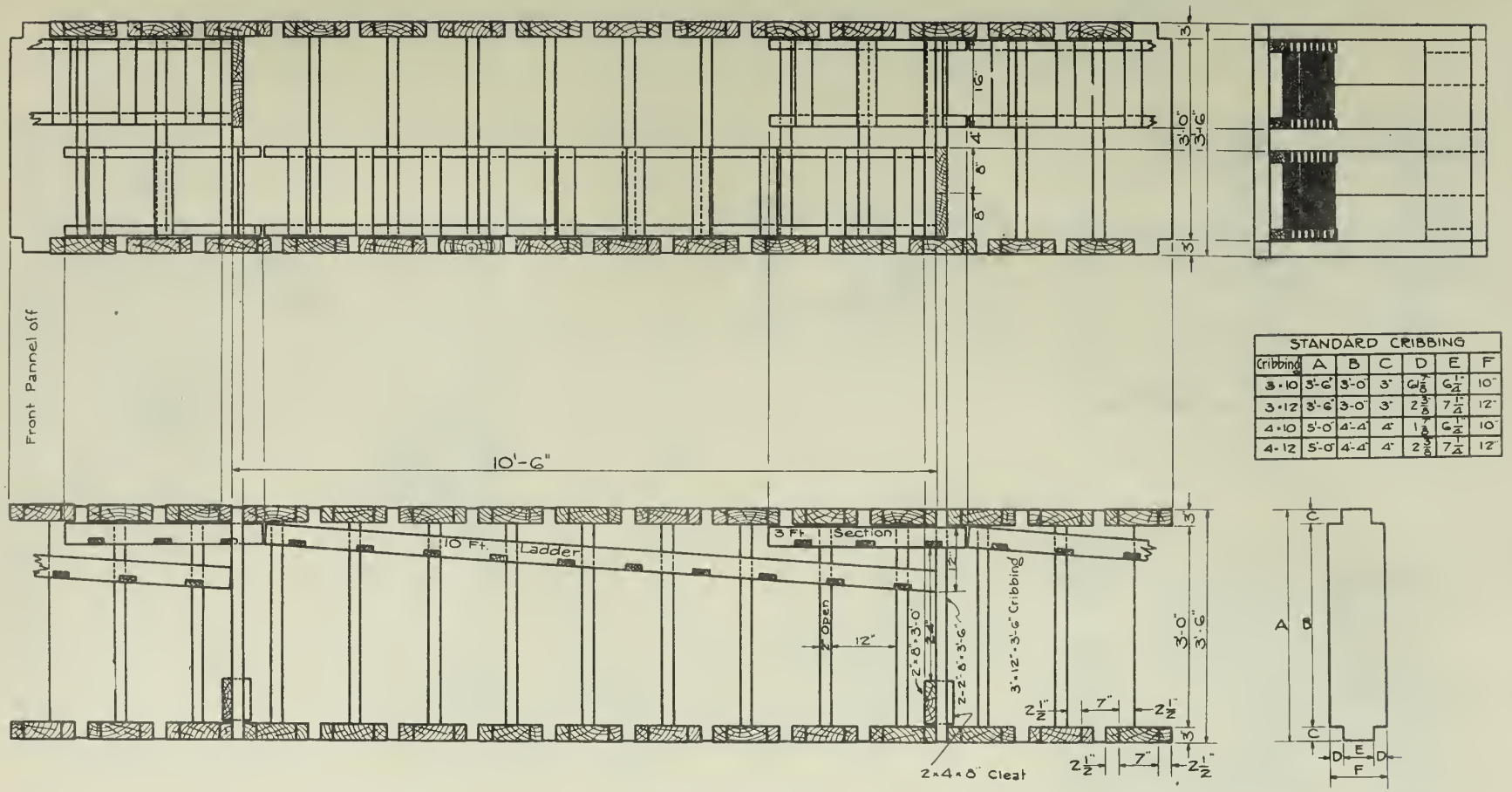
By July, 1928, when regular production started in the Colorada, approximately 1,500 feet of shaft and 900 feet of winze had been sunk and 3,900 feet of raises and 36,000 feet of drifts had been run. In July and August five stopes came into production, one shrinkage stope above the 500 level, a square-set and a cut-and-fill (rill) above the 600 level, and two cut-and-fills above the 700 level.

Since that time development and stoping have gone on together. The 1,000 and 1,200 levels will be motor-haulage levels; the former is now nearly completed. It should be borne in mind, when inspecting the cost tables at the end of this paper, that development has advanced faster than mining. It is estimated that the average cost per ton for development, over the life of the ore body as now calculated, will be less than half the present figure.

Shaft-Sinking. - Cananea shaft-sinking practice is well illustrated by the work in the Colorada shaft from the middle of 1926 to the end of 1929.

With two or three rounds broken at the surface, the collar set was placed and a standard wooden sinking headframe erected. A compressed-air hoist was used at first to operate the sinking cage but was soon succeeded by a 112-hp. electric double-drum hoist and then by the cage hoist now in use. In addition, when the shaft reached the 500 level, a hoist room was cut on the 450 level, and another 112-hp. electric hoist was installed to handle the sinking operations and release the surface hoist for other service.

Figure 9 shows the round used in the 4-compartment section of the shaft, below the 500 level. Six 65-pound sinking drills and 1-inch hollow-round steel were used for drilling. Drill steel is the same for all mining operations as regards bits and lengths. The common form of cross bit having a 5



STANDARD CRIBBING						
Cribbing	A	B	C	D	E	F
3·10	3'-6"	3'-0"	3"	6⅛"	6¼"	10"
3·12	3'-6"	3'-0"	3"	2⅝"	7¼"	12"
4·10	5'-0"	4'-4"	4"	1⅝"	6¼"	10"
4·12	5'-0"	4'-4"	4"	2⅝"	7¼"	12"

FIGURE 7. – STANDARD CRIBBING AND STANDARD 3'×3' SAFETY MANWAY

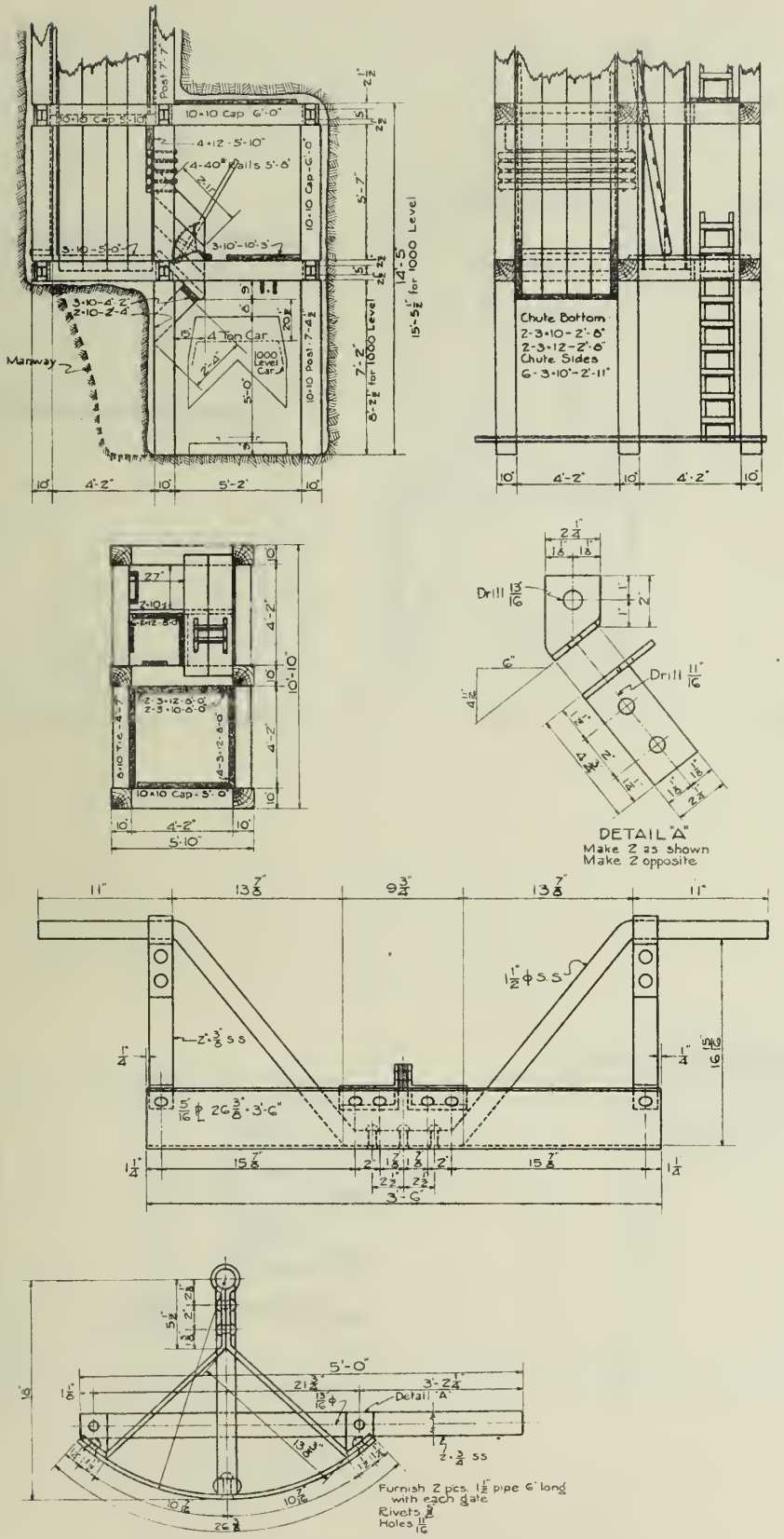


FIGURE 8. – STANDARD SQUARE SET RAISE AND CHUTE
800 LEVEL, "LA COLORADA" MINE



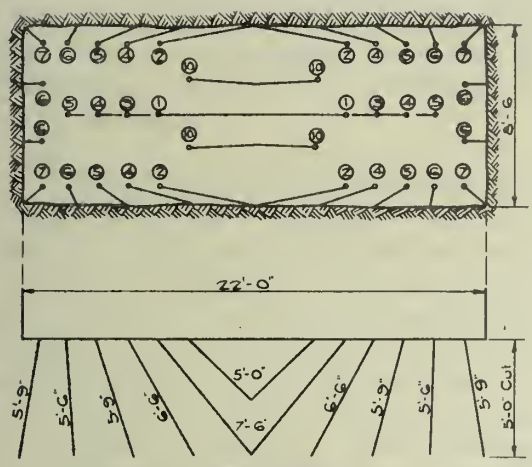


FIGURE 9.- FOUR COMPARTMENT SHAFT, ROUND

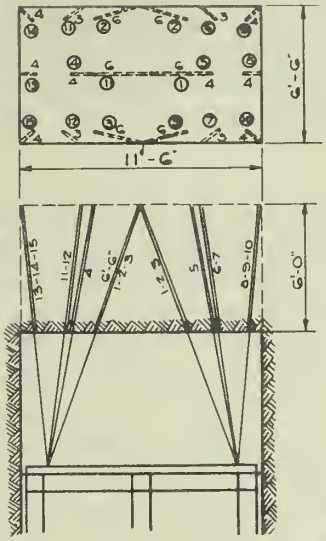


FIGURE 10. - STANDARD RAISE ROUND

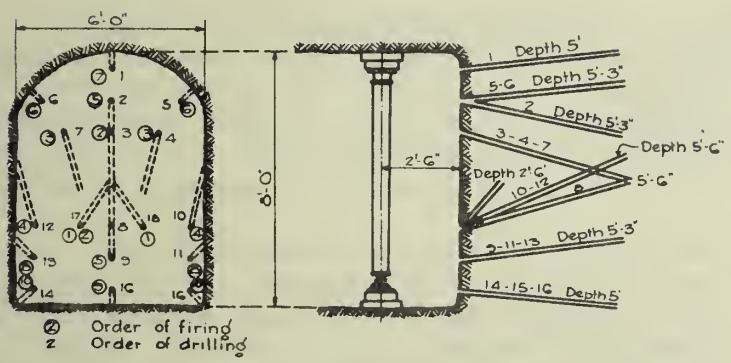


FIGURE 11.- 5-FOOT DRIFT ROUND FOR HARD GROUND

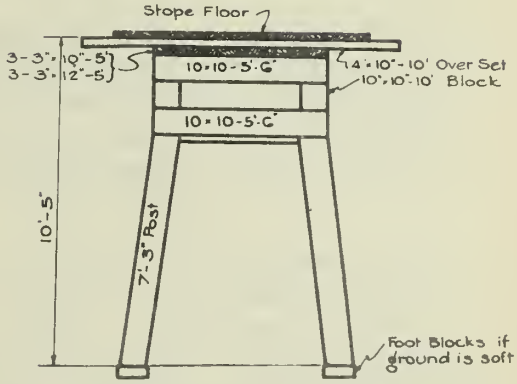


FIGURE 12.-SECTION THRU DRIFT OR CROSSCUT



and 15° double taper is used. Starter bits are 2 inches in diameter; the following bits decrease by one-sixteenth inch in gage. Starters are 2 feet 9 inches long and succeeding pieces are each 18 inches longer. All steel is hollow, and no dry drilling is allowed underground. Electric blasting with delay action detonators was used throughout the sinking of this shaft. Above the 600 level the usual Cananea practice of shoveling the rock broken by the blasts into a 1-ton car was followed, except for a short period when a sinking bucket was used. The bucket was dumped over a slide into 2-ton motor cars on the 500-level station. A special light single-deck cage, with extension guides permitting the cage to be lowered onto the bottom, is used for mucking into cars in shaft sinking. The rock is shoveled directly into the car through the open sides of the cage.

The publication of an article describing a shaft-sinking device used at the Elm Orlu mine at Butte[4] led to the adoption of an auxiliary mucking skip --a shallow steel box, open on top and at one end, hung by a bail which allowed it to be easily dumped. This was hoisted by a single-drum air hoist placed on I-beams in one of the center compartments. The skip, which held one carload, was lowered onto the muck pile and filled; the cage and car were then dropped to the bottom, the skip was hoisted a few feet, and the muck was dumped into the car. The advantage of the skip is that it can be placed to best advantage, as, for example, when the blast throws most of the rock to one end of the shaft. It is so low that it is much easier to shovel into it than into the car or a bucket. Finally, it never leaves the bottom, so that no delay occurs while the rock is being hoisted. Its introduction gave an increase of 20 per cent in sinking speed. A maximum of 107 tons was mucked out in seven hours, and never over 12 hours was needed to clean out the usual 6-foot round.

The Colorada shaft timber set has already been shown (fig. 5). An interesting detail is the use of a bridge piece outside the wall and end plates to hold the lagging in place. This is put in place on the timber before it leaves the framing shed. Lagging is placed very rapidly and needs no blocking or nailing. When once in, it can be removed only by being cut out. To use the bridge piece the shaft need be made only slightly larger.

No fixed cycle of operations was followed. If a crew came on shift and found the shaft mucked out and timbered, they would drill and blast in four hours and spend the rest of the shift mucking. The following shift would at least finish mucking. The third shift, if the ground would stand without timbering, as was often the case, would drill and blast again. When it was possible to hang and block two sets in one shift, this was done.

The shaft crew consisted of eight men, including a jigger boss, in the shaft, one top lander, and one hoist engineer. The direct sinking cost for four compartments from the 500 to below the 1,300 level, a distance of 1,027 feet, was as follows: Labor, $43.46; supplies, $27.26; power, $2.44; sundries, $1.02; total $74.18. This does not include stations, pockets, preparatory work, or hoist installations.

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4 - Drullard, H. R., Shaft Sinking To-day at Butte. Eng. and Min. Jour., vol. 123, Feb. 26, 1927, p. 357.

In 1925 the Capote five-compartment shaft was deepened from the 1,400 to the 1,600 level, a total distance of 212 feet. The sinking cage and car were used in this work. The crew consisted of six shaftmen, one hoist engineer, one pumpman, and a boss. Careful and detailed costs were kept on this work, of which Table 1 is a summary.

Table 1. - Cost of sinking Capote shaft from 1,400 to 1,600 level

July to October, 1925,
Footage, 212 feet.

Preparations for sinking shaft	Shifts	Amount	Total	Per foot
Shaft and 1,400 level station:				
Labor	194-1/2	$542.12		
Supplies	-	302.87		
			$844.99	-
Hoist repairs and installation:				
Labor	72	205.70		
Supplies	-	198.41		
			404.11	-
Air-pump repairs:				
Labor	6-1/2	20.63		
Supplies	-	38.96		
			59.59	-
Pipe lines:				
Labor	34-1/2	92.20		
Supplies	-	346.61		
			438.81	-
Sinking-cage repairs:				
Labor	11	33.00		
Supplies	-	8.95		
			41.95	-
Hoist operation:				
Labor	18-1/2	48.50		
			48.50	-
Miscellaneous:				
Labor	2	3.75		
Supplies	-	12.48		
			16.23	-
Sundries:				
Shops and planing mill	-	220.04		
			220.04	-
Total preparatory work	339	-	$2,074.23	9.80

Table 1. - Cost of sinking Capote shaft from 1,400 to 1,600 lever--Continued

Sinking shaft:	Shifts	Amount	Total	Per foot
Direct sinking:				
Labor: Shaftmen	1572-1/4	$6,289.00		29.73
Hoistmen	274	858.00		4.05
Pumpmen	266	1,064.00		5.03
Bosses	267	1,959.68		9.26
Bonus	-	1,116.95		5.28
			$11,287.63	53.36
Explosives and carbide:				
410 sticks 7/8" powder	-	20.30		0.09
9,354 sticks 1-1/8" powder (106 sticks per 50-lb. box)	-	746.71		3.53
1,765 delay detonators	-	315.43		1.49
1,375 pounds carbide	-	82.45		0.38
			1,164.98	5.50
Gum clothes	-	485.23		
			485.23	2.29
Supplies	-	71.38		
			71.38	0.33
			13,009.22	61.50
Hanger bolts:				
Labor	23-1/2	45.75		0.21
Supplies	-	245.69		1.16
			291.44	1.37
New drills, steel, and hose:				
Supplies	-	1,688.21		-
			1,688.21	7.98
New tools:				
Labor	12	33.65		0.15
Supplies	-	165.76		0.78
			199.41	0.94
Hoist repairs:				
Labor	11-3/4	39.37		0.18
Supplies	-	20.38		0.09
			59.75	0.27
Air-pump repairs:				
Labor	24-1/2	83.37		0.39
Supplies	-	199.37		0.94
			282.74	1.33

Table 1. - Cost of sinking Capote shaft from 1,400 to 1,600 level -- Continued

Sinking shaft:	Shifts	Amount	Total	Per foot
Pipe lines:				
Labor	32	90.14		0.42
Supplies	-	892.60		4.22
			982.74	4.64
Drill repairs:				
Labor	6-1/2	18.75		0.08
Supplies	-	213.30		1.00
			232.05	1.08
Shaft doors:				
Labor	9	25.87		0.12
Supplies	-	18.70		0.08
			44.57	0.20
Blasting switch and lights:				
Labor	8-1/2	24.50		0.11
Supplies	-	26.79		0.12
			51.29	0.23
Tool and drill sharpening:				
Labor	48	117.54	-	0.55
Lowering sinking pump:				
Labor	20	59.37	-	0.28
Timber framing and material:				
Labor	385	955.84		4.51
Timber: Sets, 69,269 bd. ft.		1,901.31		8.98
Lagging, 16,330 bd. ft.		564.53		2.66
Bulkheads, 5,964 bd. ft.		198.12		0.93
Miscellaneous (ladders, guides, blocks, wedges, etc.)		243.47		1.15
Nails		101.89		0.48
Tools		6.18		0.02
			3,971.34	18.77
Framing 10 sets at Ronquillo framing shed:				
Labor	64	207.37		0.98
Supplies (tools, nails, bolts, timber)		1,417.59		6.70
			1,624.96	7.68
Air and power:				
Air for drills, 820 machine hours		324.28		
Air for pumps, 961 pump hours		378.57		
Power for hoist, 18,952 kw. h.		308.35		
			1,011.20	4.78

Table 1. - Cost of sinking Capote shaft from 1,400 to 1,600 level--Contined

Sinking Shaft:	Shifts	Amount	Total	Per foot
Sundries:				
Shops and planing mill	-	303.84	-	1.43
Total sinking	3,013-1/2	-	23,929.67	113.14
Total preparing to sink ..	339	-	2,074.23	9.80
Grand total	3,352-1/2	-	26,003.90	122.94

Statistics

Shaft section (4 compartments 4'0" by 5'0", 1 compartment 5'2" by 5'0"	28.0 by 8.5 feet
Footage	212 feet
Days sinking (3 shifts each)	87
Average footage per day	2.43
Rounds drilled	43
Depth per round	4.93
Holes per round	40
Sticks of 1-1/8-inch gelatin per round	223
Pounds powder per round	105.42
1-ton cars (20 cu.ft.) hoisted	5,022
Tons per foot	23.7

Shaft Repair. - The Cananea-Duluth mine had been idle for eight years when preparations for reopening it were started in January, 1929. The mine was flooded to above the 300 level, and the timber in the main or No. 3 shaft from the surface to the 600 level was rotted so that it had to be replaced. In addition, the same section of the shaft had only three compartments and was to be enlarged to four.

Previous experience had shown that pumps were unsatisfactory for removing the strongly acid water. Therefore a wooden headframe was erected over the old No. 2 shaft, a double-drum, 112-hp., electric hoist was set up, and 450-gallon bailing skips were installed in the shaft. Using only 400 gallons as the skip capacity, to allow for spillage, the power consumption for the first month, when the water was being hoisted an average distance of 335 feet, was 2.5 kw. h. per 1,000 gallons. During the next two months the bailers handled 300,000 to 400,000 gallons per day. In the second month, when the average distance hoisted was 440 feet, the average amount bailed was 278 gallons per minute. The corresponding figures for the third month were 500 feet and 256 gallons per minute. Extending the guides and repairing the shaft and equipment caused a delay of four to five hours each day. At a little over 500 feet, the load on the hoists was reduced by cutting a hole in the bailer and changing its capacity to 320 gallons.

The first step toward retimbering the No. 3 shaft was to build a concrete collar. The old timber was torn out for 20 feet below the surface, and the extra compartment was taken out for about 30 feet down. The latter work had to be done by hand-drilling, as the compressor was not yet in operation. At a depth of 20 feet a hitch was cut all around the shaft, 2 feet deep, 2 feet high at the back and 4 feet high at the front. Temporary bearers were placed below it, forms built, and a concrete collar was poured.

Air was now available and the work of enlarging and retimbering the shaft was started. The round for the extra compartment consisted of seven vertical 6-foot holes blasted with four sticks each of 40 per cent gelatin and with fuse and caps. Two heavy sinking drills were used, and about an hour was usually needed to drill, load, and blast the round. After blasting, the old timber set was taken out. Any pieces that could be handled easily were hoisted to the surface, and the rest was allowed to fall to the 100 level. This resulted in frequent jams in the shaft, which were usually removed by blasting from below. A bulkhead had been built on the 100 level. All rock from the new compartment, as well as the old timber, fell to this level. The rock was trammed to old stopes and dumped for fill. The amount of old timber falling down made it impossible to use a chute, and all rock had to be shoveled by hand into the cars. As work progressed, the bulkhead was moved from level to level, so that it was never more than 100 feet below the new timber. Entrance from the surface to these levels was by way of a series of raises.

At first the working platform for the retimbering was simply laid on top of the old timber. A point was soon reached, however, where the old timber was too weak even to support the platform and often fell out for 40 or 50 feet below. Then a platform was made by hanging 20-foot 8 by 8 inch stringers below the last set and laying a floor of 3 by 10 inch plank on them. The 8 by 8's were hung on 3/8-inch iron chains 18 feet long with grabhooks at both ends. The chains passed around the corner posts, outside the bridging of the wall plates, and down to the platform stringers. Thus, the chains were out of the way of the set to be put in. The platform was lowered by ropes a set at a time as needed. When blasting, the end of the platform over the blast was raised a few feet and the planks were removed or piled at the far end. Men going down on the bench of the new compartment to drill always were safety belts.

As when sinking the Colorada and other shafts in the district, the timber sets were framed and completely assembled at the framing shed, checked over, numbered on each piece, and the joints lettered before the set was sent to the mine. This prevented misfits and simplified the delivery of the proper pieces to the shaft bottom. As it was often necessary to timber closely on account of the blocky nature of the volcanic tuff in which the shaft is sunk, and as there was usually not enough room below the new timber to swing a wall plate into place, the practice was followed of temporarily leaving out one divider on the bottom set. The wall plates were left unblocked opposite that piece. When the set below was in place, a screw jack was used to spring the wall plates apart far enough to drop the divider into place.

From the bottom to 40 feet above the 600 level the shaft was already divided into four compartments and the timber was in good condition. To avoid

damaging this part of the shaft, after the 500 level was passed, three compartments were bulkheaded over 40 feet above the 600 level, leaving open only the one next the fourth compartment. This open compartment was lined to form a chute so as to allow rock to drop to the 600 level without damage to the shaft.

When the water level was reached the work was interrupted. Bailers were then installed in the shaft and operated until the water was down to the 800 level, since which time the bailers in No. 2 shaft have been run a few hours a day to keep the mine dry. Bailing in No. 3 shaft presented difficulties due to the presence of old timber and partly broken sets which prevented the bailing skips from being lowered under water. Heavy tongs similar to ice tongs proved useful for grappling and pulling out submerged timbers. The bailers required extension guides into the water, and often the stumps of the old guide lag screws prevented the proper placing of regular wooden guides for this purpose. Skeleton guides were made of long pieces of strap iron bent double and held by two or three cross pieces. The free ends were lapped onto a short length of wooden guide which was drilled to be bolted onto the dividers. These guides were moved down from set to set, and as they were open front and back the old lag screws did not interfere with their placing.

Enlarging and retimbering were resumed on August 2 and finished to 40 feet above the 600 on August 25.

From February 22 to April 26, 453 feet of shaft, including four stations, was enlarged and retimbered, averaging 7.1 feet per day. From August 3 to August 22, 158 feet was completed, averaging 7.9 feet per day. Very commonly two sets and often three sets were put in in a day.

The shaft crew consisted of five shaft men and a jigger boss, three muckers below, a top lander, a hoist engineer, a compressor man on three shifts, and a foreman on one shift. During the entire difficult and dangerous job, not one man received a serious injury.

Drifting. - All drilling in drifts is done with Leyner-type machines mounted on 3-inch columns with cross arms. A universal column clamp is replacing the old-style cross arm. This clamp consists of three castings which serve to clamp a length of standard heavy 3-inch pipe at right angles to the drill column. The pipe can be slid back and forth through the clamp and may be of any desired length. This provides a more flexible drill set-up and permits drilling a wider face than was possible with the former type of cross arm.

From time to time drill instructors have been employed to teach the miners the best methods of drilling and breaking their rounds. As a result, although no standard round is insisted upon, most of the drift men use a round like that shown in Figure 11. This round is designed for 6 by 8 foot drift in hard ground.

Forty and sixty per cent strength gelatin dynamites are used in drifting, with No. 8 caps and fuse. Stemming is used. This is made by filling paper cartridges with screened table tailings from an old pond; the stemming is slightly dampened before being sent underground.

Drift crews consist of a miner, usually working on the day shift, and two muckers working on the opposite shift. The usual practice is to drill and blast on one shift and muck out on the other. Timbering, when necessary, is done by the miner before setting up and drilling. Only rarely is the ground of such nature that overhead booms or spiling are required, as in some places in the Capote and Elisa mines. Frequently timbering is done on a separate contract after completion of the drift.

Shoveling sheets are laid before blasting. The broken rock is shoveled into 1-ton cars and trammed to the shaft. A mechanical shovel operated by compressed air was tried for drift work, but without success.

Raising. - Figure 10 shows a typical raise round. Hand-rotated stopers are used for this work, with 1-inch quarter-octagon hollow steel. Raise crews consist of a miner and helper on day shift and the same on night shift.

For long timbered raises an air hoist is set up at the bottom of the raise for hoisting timber and tools to the face. In one place on the 600 level a number of square-set raises were being driven at the same time. An air hoist was mounted on a truck and taken from raise to raise whenever timber was to be hoisted. In ordinary stulled raises, timber is hoisted by hand with a rope and tackle.

The raise men are usually required to tram their own rock to the shaft. Occasionally, if contract trammers are working in near-by stopes, they are given the raise tramming to do.

Supply System. - A crew of nippers deliveres sharp steel from the shop on the surface near the shaft to the steel racks on each level in the mine. The miners get sharp steel from the racks and return the dull pieces, which are then taken to the shop by the tool nippers. As almost all drilling is done on the day shift, the nippers work on night shift, so that each morning a supply of steel for the day's work is on hand.

Each level has a supply room where tools, drills, hose, and other supplies are kept. If a machine breaks down, the miner has only to take it to the supply room and exchange it for one of the spare machines. The tool nipper takes the broken machine to the surface and returns it when repaired. Hose is likewise repaired on the surface.

All timber is framed at the framing shed at Ronquillo and taken by the narrow-gage railroad to the mine timber yards. There it is loaded on timber trucks, hauled to the shaft, and lowered into the mine. On each level small amounts of timber are stored at points as close as possible to the working places so that contractors and miners will not waste time hunting for timber or hauling it long distances. Generally the timber for the stopes is lowered through raises from the level above. Wedges are not made into bundles but are shipped loose and stored in bins in the timber yards and underground.

The main powder magazine is at the upper end of the Capote basin. Powder is brought to the mines each day by railroad or truck. There it is loaded on the cages, distributed to the several levels, and stored in the underground

magazines. Only one day's supply may be kept in the magazines. To be hauled into the mine, the boxes of powder are packed into strong, red-painted, locked wooden boxes, which are loaded on the regular timber trucks. Fuse from 3,000-foot reels is cut and capped at the surface at each mine. A day's supply for each level is packed into special containers and sent to the underground fuse magazines. The containers are carbide cans, lined with burlap and fitted with carrying handles. Miners get requisitions for their powder from the shift bosses. Powder and fuse are carried from magazine to working place in separate sacks made from discarded flotation-cell blankets.

Timber-Treating. - All shaft timber and all timber for permanet workings, such as main-haulage drifts and shaft stations, is treated with a preservative after framing and before being sent to the mines. The treatment is for the double purpose of increasing the life of the timber and making it resistant to fire.

The timber is chiefly Oregon pine from the Pacific coast. The preservative, a yellow powder, is a mixture of triolith (90 per cent sodium fluoride, 6 to 7 per cent potassium or sodium bichromate, and small amounts of phenols) and sea salt, bought in 675-pound barrels. The latter comes from natural deposits on an island in the Gulf of Lower California.

The solution was at one time mixed in a galvanized iron tank. After about two years of service the tank wore out because of the corrosive nature of the salt solution. It has been replaced by a redwood tank coated inside and out with a heavy asphalt-base paint. The tank is 4-1/2 feet square and 6 feet deep, having an effective capacity of about 850 gallons. The tank is filled half full of water, and the water is heated by turning in live steam. Four pails, or 112 pounds of triolith, is added and stirred and dissolves at once. Then 300 pounds of the sea salt is dumped in, the tank filled to the top with water, and the steam left on to heat the mixture and give some agitation. The salt dissolves in about two hours. The solution then has a specific gravity of about 10° Baumé.

There are two treatment tanks, sunk level with the ground near the framing shed. They are made of reinforced concrete 8 inches thick and have one side wall in common. Inside dimensions of both are the same; 6 feet wide, 24-1/2 feet long, and 6 feet deep. A 20-gallon-per-minute electric centrifugal pump is piped to transfer solution from one tank to the other. Steam lines of 2-inch pipe are supported on wooden blocks about 2 inches above the floor of the tanks, one tank having four and the other six lines lengthwise. Originally, small coils of 1-inch pipe with standard threaded fittings were used. These soon developed leaks due to corrosion, and were very difficult to repair. The present pipes are welded to avoid this trouble. Steam at about 40 pounds pressure is piped from the smelter power plant. Above the pipes, and likewise supported on wooden blocks, is a grating of old rail laid across the tank about 6 inches above the floor and spaced a foot apart.

Timber is loaded in and out of the tanks by hand. Plank and large pieces are laid evenly, with small strips of scrap lumber for vertical spacing, so that the solution can circulate freely around and through the pile. Blocks

and wedges are dumped on a floor of boards and covered with boards on top. The wood is kept from floating by wooden bars wedged into slots in the concrete walls a few inches below the top of the tank.

The loaded tank is covered with a double layer of plank, and steam is turned into the pipes for about 12 hours, usually overnight. This dries and heats the wood, drives out some of the pitch, and makes it absorb the solution more rapidly. In the morning at 7 o'clock when the men go to work, the solution from the other tank, at a temperature of about 100°F., is pumped onto the new timber. After three to five hours it will have heated to 190 to 200°. The desired treating range is from 194 to 198°. Above this the loss of water is too rapid. When the solution is first applied, it is tested and brought to 5° Baumé, either by adding water or the strong fresh solution. The test is repeated after a few hours and enough liquid is added to keep the wood submerged. After eight hours, at 4 o'clock in the afternoon, the steam is turned off and the timber allowed to soak in the slowly-cooling bath until morning. The solution appears to penetrate the wood about one-fourth inch.

The capacity of each tank is approximately 4,500 board feet of ordinary mine timber. As much as 107,000 board feet has been treated in a month. During the first half of 1929 about 380,000 board feet of timber, or one-tenth of the total mine consumption, was treated; 3,950 pounds of triolith and 11,200 pounds of salt were consumed.

Stoping Methods

At one time or another Cananea has offered examples of almost every known mining method.[5]

In 1910 eight or more mines were in operation, producing ore by a half-dozen distinct methods. At Puerticitos a low-grade ore in garnetized limestone was being mined by surface benching. Primary breaking was by means of 22 to 24 foot hand-drilled holes, blasted with black powder. A large proportion of waste was sorted out to produce a 4 per cent ore, and 75 to 80 per cent of the material broken was discarded. The mining efficiency was 4-1/2 tons of ore and waste per man-shift. Square-setting was in use, but chiefly as an auxiliary to other methods, accounting for not over 5 per cent of the total production. Its cost was relatively high, varying from $1.20 to $1.40 per ton of ore in the chutes for mining and timbering only. Top slicing held first place as a producer. This differed little from the method as used to-day. It gave a low stoping cost of $0.60 to $0.70 per ton of ore in the chutes. A combination shrinkage and caving system was used under certain conditions, which consisted of shrinkage stopes and the caving of pillars. The stopes were carried up to capping or stoped ground above, after which the pillars crushed or were blasted out. No timber was needed, and a cost of $0.40 to $0.50 per ton was attained. A cut-and-fill method was in use in the hard ores of the Elisa mine, where the stopes were sometimes as large as 75 by 100 feet. The method differed from the modern cut-and-fill method in that no floor was used and in that sorting was necessary, as about 50 per cent of the rock broken

5 - Elsing, M. J., Mining Methods Employed at Cananea, Mexico. Eng. and Min. Jour., vol. 90, Nov. 5, 1910, p. 914; vol. 90, Nov. 12, 1910, p. 963.



FIGURE 13.- HORIZONTAL PROJECTION OF STOPES IN CAPOTE BASIN

was waste. To prevent the loss of fines the ground was broken as coarsely as possible. Finally, what was known as pyramid stoping was carried on in the Veta Grande. This was a combination of square-setting and shrinkage stoping. Blocks 40 by 50 feet were laid out by square-set drifts on the sill floor. From the drifts a double row of square sets was carried up all around the block. The 30 by 40 foot solid core in the center was mined out by shrinkage stoping, the excess ore being drawn out through the square-set chutes. Waste was necessarily mined and trammed away. When all the blocks in a given section had been mined, the broken ore remaining in them was drawn out. Pyramid-shaped piles of ore would still be left in the middle of each stope, and these gave the system its name. They were removed by running drifts through the centers of the blocks, with one or more chutes to tap the broken ore above. The cost of stoping by this method was 80 to 90 cents per ton.

At present the Capote, Veta Grande, and Colorada are using shrinkage, cut-and-fill (both horizontal and inclined), square-setting, top-slicing, and undercut-caving methods. This diversity of methods is chiefly due to the great natural variation in ground conditions and ore occurrence. Occasionally previous development or mining, plans for future operations, or the security of shafts or surface plant are the deciding factors in choosing methods.

Figure 13 shows that the ore bodies worked by the Capote and Veta Grande are very irregular in size, shape, and distribution. In this area, moreover, the wall rock is generally weak. Square-setting and top-slicing, consequently, have accounted for most of the tonnage from these mines. These methods, however, are better described in connection with the Colorada. The Capote is using, in one section, an undercut caving method, which will be described.

The ore and walls in the Colorada are of such nature that little or no support is necessary when mining. On the 600 and 700 levels, where the ore shoot is in effect a vein (fig. 4), long filled rill stopes were laid out as offering the cheapest method consistent with safe working conditions and complete extraction. Some of these are being converted to horizontal cut-and-fill. Below the 700 level the ore ring becomes much thicker, and just below the 900 the waste core disappears completely. It was evident that if the cut-and-fill method were to be used, pillars would have to be left to support the walls and then removed by some other method, either by top-slicing, which might allow a general subsidence of ground above, or by the Mitchell slicing method, attempting to fill so as to prevent any general caving. The former course promised more certain recovery of all the ore. Therefore, it was decided to use top-slicing for removing pillars from the 800 to the 1,000 level. Below the 1,000 it is planned to top-slice the entire ore body. However, if production requires the opening of stopes on these lower levels before the top-slicing of pillars above is finished, cut-and-fill stopes will be laid out as described with pillars between to be mined by top-slicing.

Near the shaft and between the 500 and 600 levels, although the walls are solid, a part of the ore is so shattered and blocky that square-setting, with small sections and close filling, was the only possible method for use. This area was started as a square-set stope. A small section adjacent to it was started as a rill cut-and-fill, but was likewise found unsuitable for this

method, which was changed to the square-set.

Above the 500 the ore was of relatively low grade and the vein narrow. Some of the first production from the Colorada came from the shrinkage stopes. A disadvantage of shrinkage stoping in this case is that the ore after remaining in the stope for a short time oxidized so as to interfere seriously with flotation. This is generally true here where the ore occurs in diorite porphyry. At present, the material being drawn from this stope is of very low grade, and a large part of it is being used for waste fill in the stopes below.

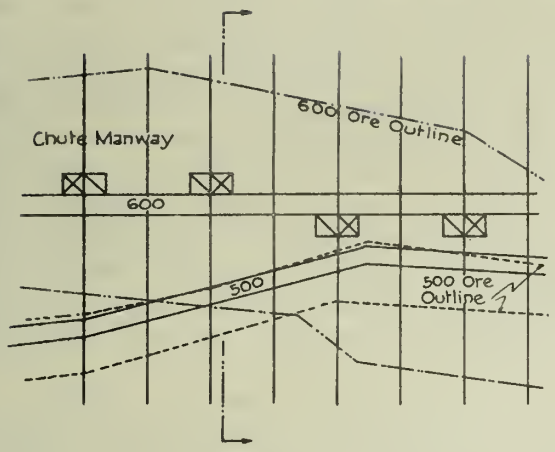
Filling is an important phase of both square-setting and cut-and-fill stoping. As much as 5,000 tons of waste is sometimes required for filling one floor of a cut-and-fill stope. Most of this waste at present comes from two surface glory holes, but a small amount is from development work. One glory hole is over a 500 level tunnel which extends through a small ridge between the Colorada ore bins and the Kirk shaft. The other is above the 500-level ventilation adit.

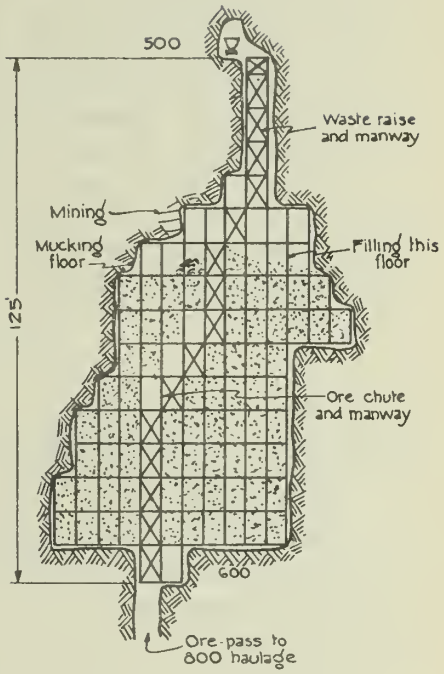
Table 2 shows some comparative stoping costs for Colorada stoping methods.

Table 2. - Partial stoping costs, Colorada Mine, August and September, 1929

	Square-set	Horizontal cut-and-fill	Rill cut-and-fill
August, 1929:			
Tonnage	9,353	15,577	8,539
Labor	$0.6101	$0.4829	$0.5421
Explosives	.0787	.1054	.1689
Timber	.7156	.2252	.2142
Partial totals	1.4044	0.8135	0.9252
September, 1929:			
Tonnage	8,980	18,483	8,775
Labor	$0.5332	$0.5146	$0.4482
Explosives	.0691	.1562	.1110
Timber	.5436	.2789	.3509
Partial totals	1.1459	0.9497	0.9109

These figures are for ore delivered to chutes on haulage levels, and include breaking, timbering, and mucking. Stope bosses are included in "labor." Supplies, drills, and air are not included, as these are distributed evenly to all ore, amounting to a total of 17 cents per ton in August and 20 cents in September. Filling is charged similarly and amounted to 17 cents per ton in both August and September.





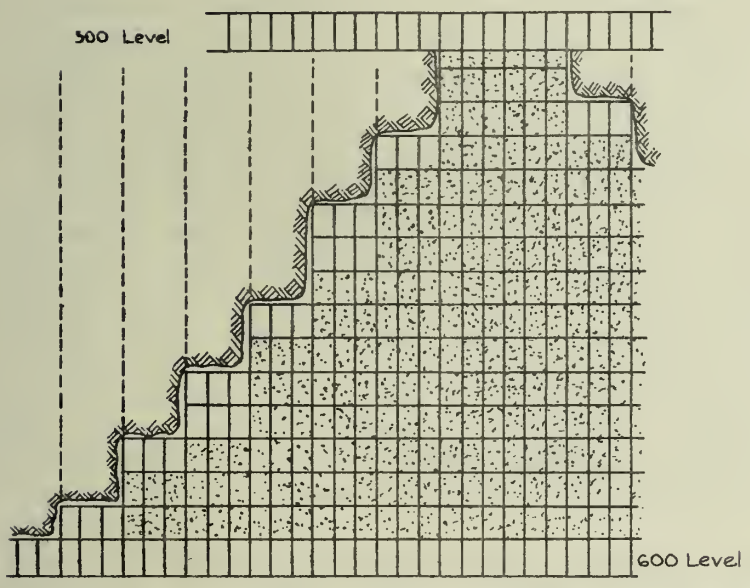


FIGURE 14.- SQUARE SET STOPING

SCALE: FEET

0 10 20 30 40 50 60 70

Square-setting. - The square-set stopes lie between the 500 and 600 levels, extending over a length of about 200 feet. The vein averages 60 feet wide on the 600 level and 20 feet on the 500 level (fig. 14). A drift was run in the center of the vein on the 600 level and timbered with sill floor square-sets. From it two-compartment six-post raises were driven to the 500 level at intervals of 30 feet along the vein. These raises are on the dividing line between the 15-foot stope sections and are offset as they go up to keep them as nearly as possible in the center of the ore. One compartment is a manway and the other a chute. The raise serves for entrance for men, lowering timber and tools into the stopes, dropping waste fill, and extracting the ore. Each raise serves two sections. Eleven feet above the floor of the 600 level, a 15-foot section is mined across the ore body, in both directions from the raise, and is timbered with standard 10 by 10 inch square-sets. Posts are stood on the solid and the floor is fitted around them. The floor consists of a single layer of 2 by 10 or 2 by 12 inch plank 10 feet long, laid on sills of doubled 2 by 10 inch plank. The sill is not taken out because to do so would make the maintaining of the drift difficult. Eleven feet, the usual height of a top-slice floor, is left because top-slicing is to be used to remove the sill after square-setting is finished. The ground over the drift but below the stope floor is taken out and a bridge cap put in (fig. 12).

Another floor is mined next by starting at the raise and blasting the ore down on the first floor where it is shoveled into the chute. The sides of the stope are next lagged inside the posts with 2 by 10 inch plank, and the posts are laced top and bottom from one side of the stope to the other with wire rope. This rope is obtained by unstranding discarded hoist cable and is made up on the surface into 40-foot lengths and coiled for convenient handling. This lacing is necessary to prevent the weight of the fill from breaking out the gob fence when the adjacent section is brought up. Fill is now run in from the level above and spread by hand or occasionally by single drum scrapers or slushers. A temporary floor is then laid over the fill. Mining and filling are now carried on together, mining on one side of the raise and filling on the other, thus speeding the work and leaving as little ground open as possible --never more than two floors. Where the vein is wide a second extraction chute may be made by lagging-off sets before filling. Adjacent sections are started and kept within two or three floors of the preceding one, as shown in Figure 14. This order is maintained strictly, to prevent the formation of high vertical brows. For the same reason lead sets are always taken on the side next the gob; often three or four sets are carried in advance of the stope to break the corner of the brow before it falls of its own weight and crushes the floor below. Very careful blocking is necessary. Wheelbarrows are generally used for spreading the fill and moving the ore to the chutes, although slushers are very satisfactory and are being used whenever available.

No tramming is done on the 600 level. When the stope raises were being driven to the 500 level, temporary chutes were built in the drift sets, and the ore from the raises was trammed to the shaft. Before stoping began, ore passes had been brought up from the 800 level, with fingers to each of the square-set stope raises, so that all the ore from these stopes is dropped to the 800 haulage level.

The square-set stopes have produced a small amount of first-class or smelting ore. No special provision is made for handling two classes of ore, however, and no sorting is done in the stopes, either for waste or for high grade ore. If the ore drawn from the 800 level happened to reach the required grade, it was taken for first class. Experience enables the bosses to estimate grades closely enough to separate first-class ore by inspection.

Square-set stope crews consist usually of a miner, a helper, and two muckers (same on both shifts). Drilling is done with jackhammers and 1-inch hollow-round steel, and four or five flat 5-foot holes are drilled to a set. Only 40 per cent gelatin powder is used. A square-set averages 200 cubic feet.

Shrinkage Stoping. - In the Colorada mine some shrinkage stoping is being done above the 500 level in a relatively low-grade ore shoot which varies in width from 10 to 20 feet (fig. 15). A drift was driven in the ore on the 500 level along the hanging-wall side of the vein. From this drift, at about 30-foot intervals, raises were driven in the ore to 40 feet above the level. At that elevation a connecting drift was run from raise to raise and silled out to the full width of the ore to form the base of the stope. Three of the raises, as shown, were driven as two-compartment stulled raises. The one in the center of the stope was pushed up as quickly as possible to the 200 level, for ventilation. Later on it also served for a time to lower supplies into the stope. No attempt was made to maintain it through the broken ore in the stope. In the raises at either end of the stope the divider stulls were hitched from wall to wall of the stope, and the manway was kept open permanently. As the stope went higher the ore diverged more and more from the vertical ventilation raise. For a time an entrance was kept from it to the stope, forming a narrow vertical slot in the hanging wall. Finally, however, the two end raises, which were carried considerably in advance of the stope, were connected to the 200 level and the center raise was abandoned.

To preserve the 200-level drifts, which now gave access to the stope, and in which a series of grizzlies was to be installed, a 35-foot pillar was left below the level and the stope started anew above it as was done on the 500 level. The upper and lower parts of the stope were connected by raises through the level pillar, and 10-inch steel rail grizzlies were built over each raise on the 200 level. This was done to eliminate the blasting of bowlders in the 500-level chutes. A large part of the production of this stope is now being used for waste fill in stopes below, and it is desirable to be able to load the trains rapidly with waste for plugging bowlders in the chutes.

Above the 200 level the stope is being carried up as before. Connection has been made to the 100 level for ventilation, and the two end raises are being driven to the surface. A large pillar will be left to protect the main 100-level drift as long as needed, and the stope will be carried through to the surface.

Drilling is done with stoper machines, the drillers standing on the broken ore in the stope. Vertical holes 8 feet deep, three or four to a row across the stope and spaced 4 feet apart, are each blasted with four to six sticks of 40 per cent gelatin dynamite.

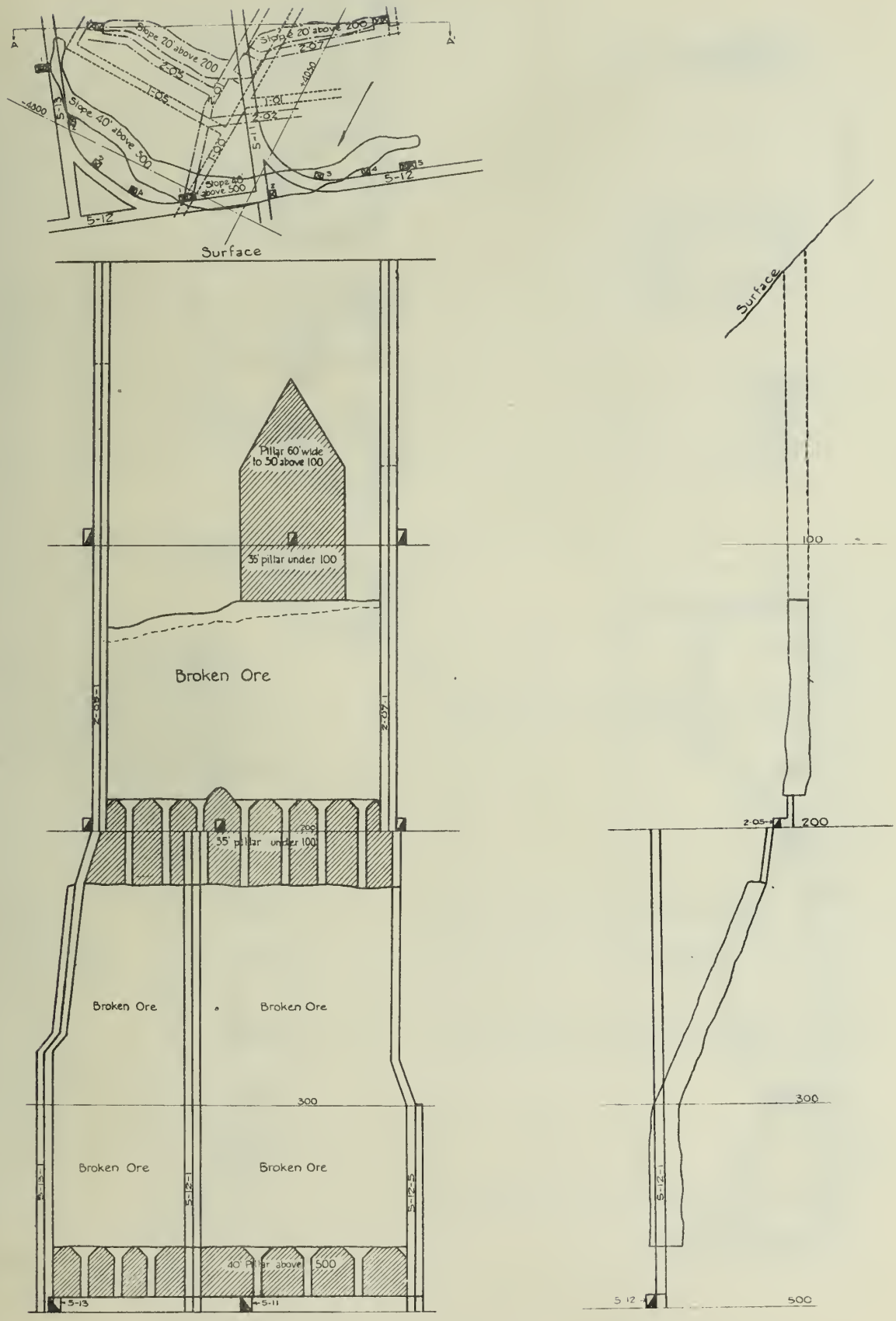


FIGURE 15.- SHRINKAGE STOPE. "LA COLORADA" MINE

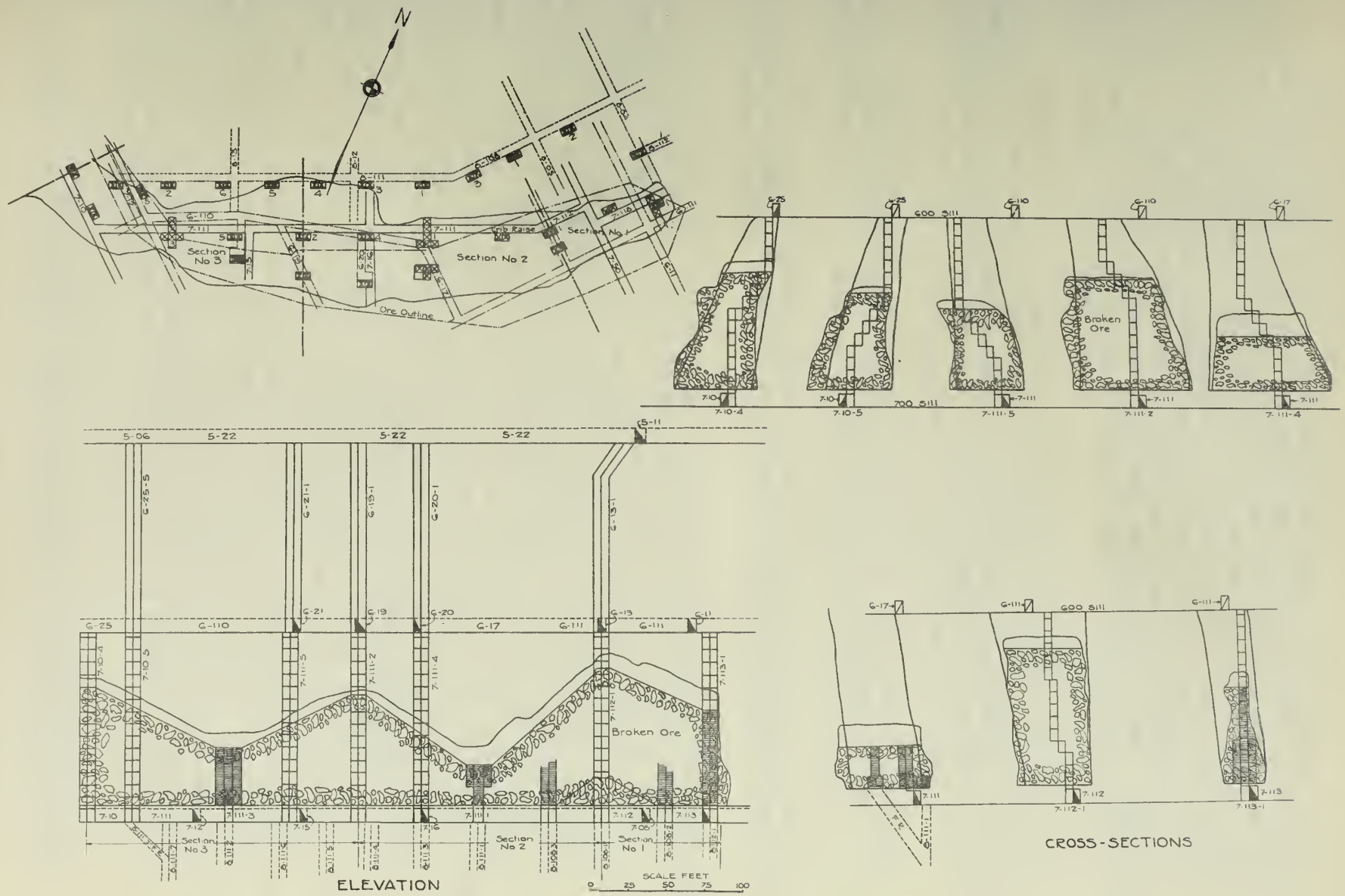


FIGURE 16. - FILLED R111 STOPE IN "LA COLORADA" MINE

The Colorada shaft is about 300 feet north of this stope. The 100 level is an adit level on the same elevation as the shaft collar. The stope has two purposes besides that of producing ore; the more important is to establish a break line on the side of the ore body next to the shaft, that will prevent the expected subsidence from extending to the shaft; the other is to provide a means of filling above the top-slice stopes to be worked from the 600 to the 900 level. The broken material in the shrinkage stope will settle down over the top slices, where otherwise dangerously large openings might form. Additional waste to keep the cut full will be obtained by glory-holing on the surface on the up-hill side of the stope.

To make a clean break line, the pillars on the 100, 200, and 500 levels will be removed when the stope is finished. The one above the 500 may be stoped for ore by cut-and-fill mining, or, like those above, may be allowed to go for waste. They will be partly or completely removed by cutting the block with drifts and raises and blasting out the remaining small pillars.

Rill Cut-and-Fill. - The rill cut-and-fill or filled rill stoping method is now accounting for 25 to 30 per cent of the Colorada production. Figure 16 shows plan and sections of one such stope on the 700 level.

Here the ore averages about 60 feet thick on the 700 level and 40 feet on the 600 level. It is hard and compact, though blocky, and stands well over the required widths.

A drift was driven about in the center of the vein and timbered with regular square-set timber. From it stulled raises were driven to the 600 level for waste-fill raises. They were continued from the 600 to the 500 level, where the waste is brought in. Meanwhile, ore passes were raised from the 800 to the 700 level at points halfway between the waste raises. The stope was next silled out for its entire length at a height of 11 feet (one top-slice floor) above the 700 rail. The ground was broken by breast stoping, carrying a face the width of the ore and pulling the broken ore to the chutes with scrapers. At this time the 2 or 3 feet of ore above the drift timbering was also taken out and a bridge cap placed over each set. A floor of 2-inch plank was laid in the stope, the extraction chutes were cribbed up to the back, and waste was run in from the 500 level and spread to within 3 feet of the back. Mining was then begun by taking out ground on both sides of the waste raises in inclined slices 11 feet thick and filling with waste after each cut, until the back of the stope was given the saw-tooth profile shown in Figure 19. Floor is laid on the slope each time after filling with waste and before starting to mine. The floor consists of 3-inch plank nailed to other plank for sills. With so wide a back the slope of the rills could not be carried steeply enough to run the ore into the chutes by gravity, because this would have formed a dangerous "belly" directly above the chute. Therefore scrapers and double-drum air hoists are used to move the ore and to spread the fill. The broken ore drops to the motor chutes on the 800 level. Above the 600 level the ore is not so wide, and it is expected that the slope can be safely increased and the use of scrapers abandoned.

Drilling is done with Leyner-type drifter machines, mounted on 3 or 4 inch columns. In advancing up the slope of the rill, 10-foot holes spaced

about 5 feet apart both ways and given the same pitch as the back of the stope break a 10-foot round across the width of the stope. Four to six sticks per hole of 40 per cent gelatin dynamite with No. 8 caps and fuse are used. Considerable trouble is experienced with the ground's breaking in huge bowlders. Blasting with 60 per cent dynamite corrects this to some extent. "Space blasting" with 60 per cent dynamite which has given good results elsewhere is being tried as another means of breaking the rock more completely. The chief fault of the present method is that the explosive is concentrated in the bottom of a long hole. In space blasting, three sticks of powder, including the primer, are tamped in the bottom of the hole. A 1-inch round stick of wood 10 inches long is inserted, the next stick of powder is tamped on top of it, and so on until enough powder has been used. The hole is then sealed with two or three sticks of stemming. As the holes are 1-5/8 to 2 inches in diameter and the spacers are only 1 inch, detonation is complete, yet the force of the explosion is better distributed and fewer larger bowlders result.

The extraction chutes are generally carried three compartments wide, and the scraper hoist is set up directly over the center compartment. Thus, the hoist can serve both slopes from the same set-up merely by swinging around on its column. At the point where the outside compartments join the center one, the heavy ore has a tendency to cut away the cribbing. To avoid this, swinging curtains made of old rail are placed over the tops of the chutes. The ore as blasted down is very coarse, so that only about two-thirds of it will go through the grizzlies without secondary breaking. Small bowlders are broken with hammers and larger ones with compressed-air concrete breakers. These machines use moils made of special alloy steel, with collars shrunk on instead of forged. The moils do not break, in spite of the severe usage. The machine weighs 79 pounds, which makes it difficult to handle under the conditions. However, it is so effective that it is considered almost essential, especially by the contractors. In addition to this work on the grizzlies, some plugging is necessary on the slope of the rill to break the largest slabs before attempting to move them with the scraper.

Stope crews consist of two muckers and two miners on both shifts. One stope boss directs the work of three or four stopes. The same crews are given contracts both for mining and filling.

Horizontal Cut-and-Fill. - The application of the horizontal cut-and-fill method is well illustrated in the stopes between the 800 and 1,000 levels. Figure 17 shows the stope layout and haulage drifts for part of the 1,000 levels. Stopes and pillars from the 900 to the 800 level are laid out exactly over those on the 1,000 level. The typical stope is 30 feet wide, from 50 to 160 feet long, and has a row of raises along the center line at 40-foot intervals. These raises, as well as the corresponding raises in the center of the pillars, are driven from the 1,000 to the 800 level as stulled raises, except that they are cribbed for about 20 feet above the rail. As the 900-level stopes are to be worked at the same time as those on the 1,000 level and as there is to be no ore haulage on the 900 level, the pillar raises are branched to tap the stope raises at the 900 level and will handle the ore from those stopes.

On the 900 level the stopes were silled 11 feet above the rail. On the 1,000 level, to protect the haulage drifts, the stopes were silled 22 feet

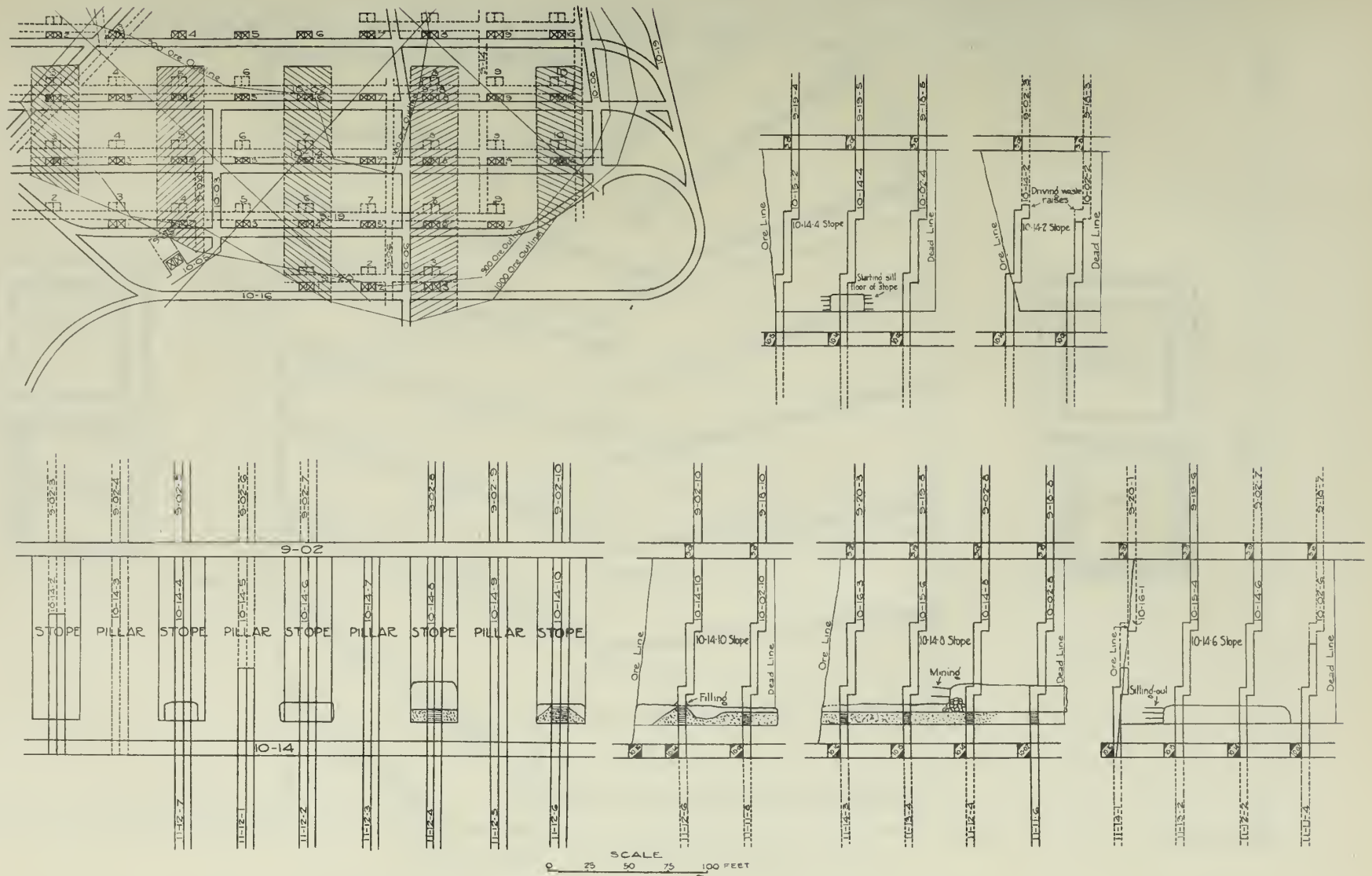


FIGURE 17. – HORIZONTAL CUT AND FILL STOPES IN LA COLORADA MINE

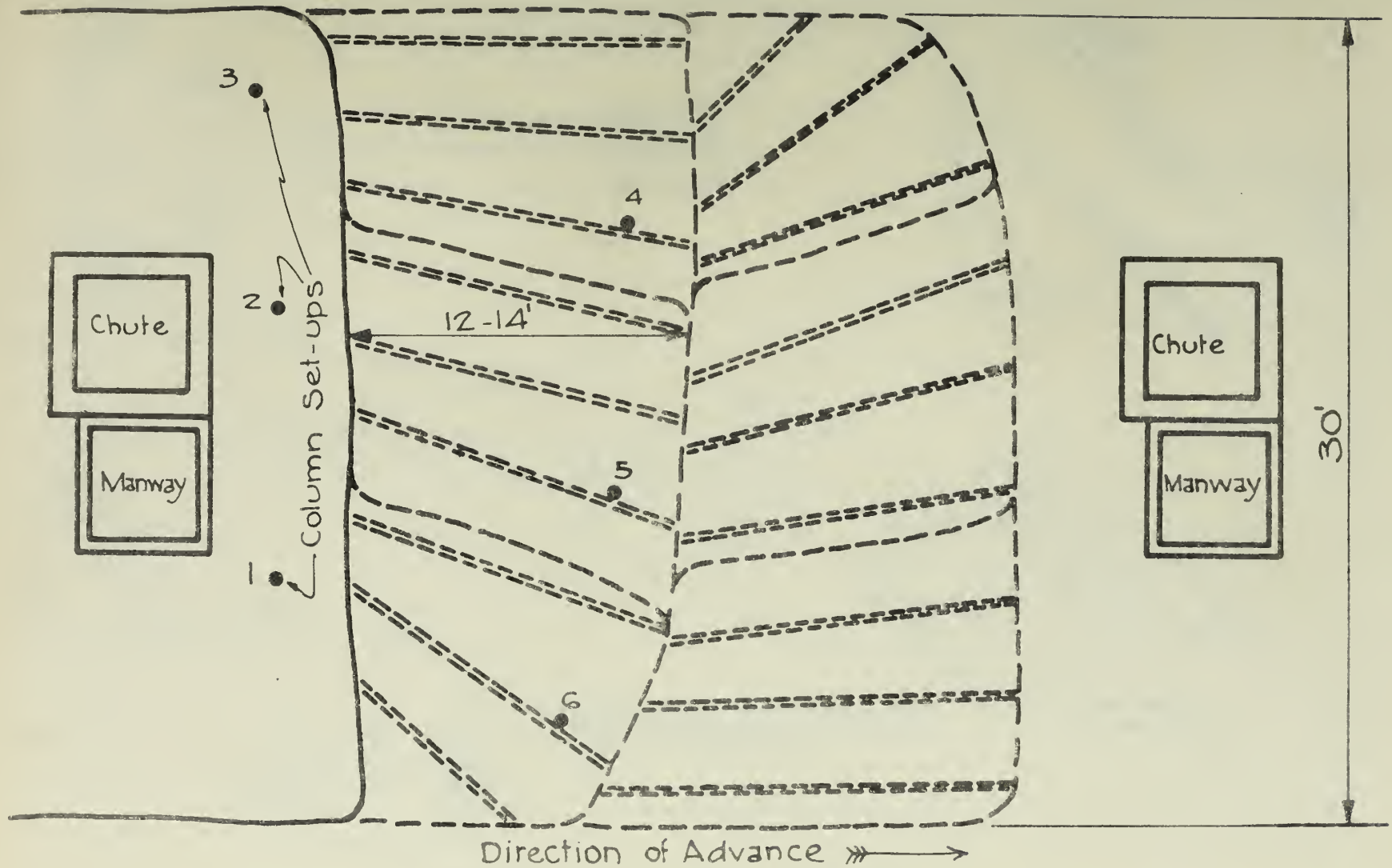
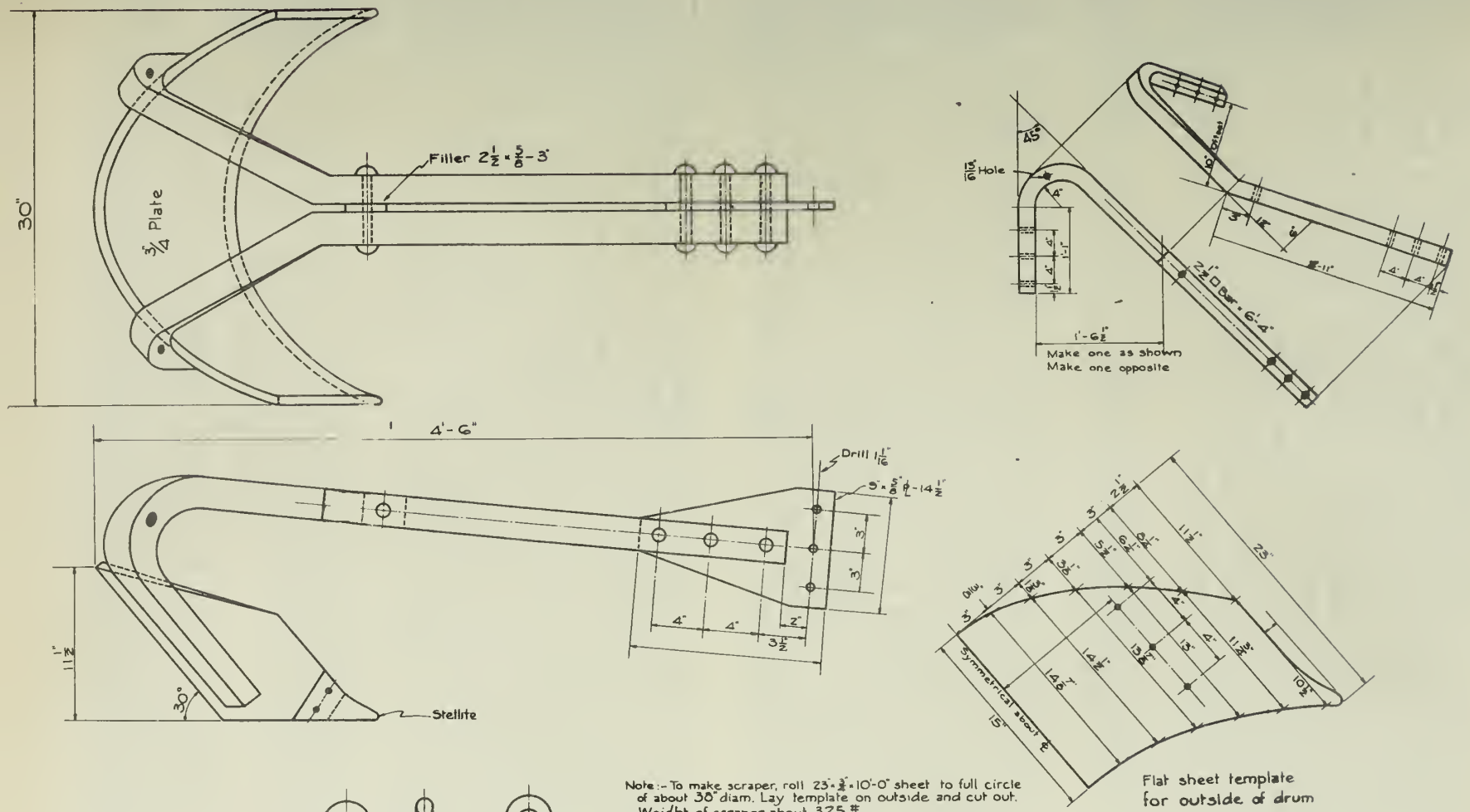


FIGURE 18.- DRILL ROUNDS FOR SILLING OUT CUT AND FILL STOPES





Note:- To make scraper, roll 23" x 3/4" x 10'-0" sheet to full circle of about 30" diam. Lay template on outside and cut out.
Weight of scraper about 325 #
Rivets 3/4" Holes 13/16" unless noted

Flat sheet template for outside of drum

FIGURE 19. - CRESCENT SCRAPER

above the floor of the drifts (two top-slice floors). From the stope raises the ground is taken out to the side line on either side of the stope, giving a face 30 feet wide and 11 feet high. This face is now advanced by breast stoping. Figure 18 shows the method of drilling. Each row of holes consists of four or five flat holes spaced 3-1/2 to 4 feet apart vertically. Some plugging and secondary blasting is done, but most of the bowlders can be broken by hand. Pavement breakers are used in these stopes for breaking large bowlders. Drilling is done with heavy drifter-type machines mounted on 4-1/2-inch columns. As in drifting, a universal clamp with heavy 4-inch pipe for a cross arm is found more satisfactory than the rigid arm formerly used. With cross arms up to 4-1/2 feet long three or four rows of holes can be drilled from one set-up.

As soon as a stope is large enough a scraper is installed to move the ore to the chutes (fig. 19). Single-line slushers have been used, but the ore breaks too coarse to be moved by a light, hand-held scraper.

The extraction raises are built up directly under the stulled raises. The manway side of the raise is cribbed throughout with 4 by 10 inch timber. The chute side is timbered with 10 by 10 inch cribbing for the first 50 feet and above that with 4 by 10 inch cribbing. Chute lining is unnecessary with the large size of cribbing, whereas when the lighter timber was used the chute not only had to be lined but the lining had to be replaced once or twice during the life of the stope; this caused delays and used almost as much timber as is used under the present plan. The chute compartment is covered with a grizzly of 30-pound rails, and the ball and rib are cut off at the ends and the base bent down to fit over the outside of the cribbing. Grizzly openings are 10 inches. The manway is kept covered with timber, leaving just enough room for a man to get through.

When the breast has advanced past a second raise, the first part of the stope is cleaned out and a floor of 2 by 10 inch plank 10 feet long, nailed to 4 by 10 sills is laid. The sills are placed parallel to the direction in which top-slicing will advance underneath. Gob fences are built around the stope, except on the waste sides. These are made of 2 by 10 inch plank, nailed to other plank in place of posts. The space between the fence and the wall is filled with ore. The grizzly rails are now removed from the chute, the extraction raise is cribbed up to within 3 or 4 feet of the back, and both compartments are tightly covered with heavy timber. Generally the outside of the raise is covered with light scrap timber to keep waste out and to give additional rigidity so that the falling waste will not shift the raise out of position.

Fill is run in through the waste raises and spread by scraper and wheelbarrow to within about 3 feet of the back. The waste from the Colorada glory hole is hard and coarse, but the smaller amount produced by the Kirk glory hole is fine and when available is used for the top foot or two of fill for convenience in leveling and laying floor.

After filling, a temporary flooring of 3 by 10 or 3 by 12 inch plank is laid on the waste and the second cut started. The method of drilling and blasting is different now from that described, as the ore has a free face underneath. Flat holes 5 feet apart horizontally and vertically, 12 feet long, and

drilled straight into the face break the ore readily; four to eight sticks of 40 per cent powder are used per hole. The only difficulty is to break the ore fine enough to handle. Where it has a tendency to come down in large slabs and blocks, small timber cribs are built up from floor to back and the ore is blasted down against their support. This tends to eliminate the hugh bowlders. In the future, space blasting will be used in these stopes, as well as in the rill stopes. Occasionally cribbing is needed behind the face to support the back. Carrying the back 2 or 3 feet higher in the center than at the sides gives an arch that reduces the trouble from loose back.

Top-Slicing. - In the Colorada mine, top-slicing will be applied to mining pillars between finished cut-and-fill stopes. These pillars will be 40 or 50 feet wide and 60 to 100 feet or more long. They will be already partly developed by drifts along their center lines and by raises from these drifts, generally at 40-foot intervals. In addition, some of these raises will have inclined branches or fingers extending toward the edges of the pillars which have been used for drawing ore from the cut-and-fill stopes on the level above. Above the 900 level the pillars will be only 40 feet wide and disconnected. Above the 1,000 level, as shown in Figure 6, they will be 50 feet wide and spaced 30 feet apart along a central 40-foot pillar which runs through the long dimension of the ore body.

These blocks will be mined in one or two sections, depending upon the number of working places desired. One or two headings, therefore, will be driven lengthwise through the pillar on the top floor, intersecting the raises or the fingers, and past the end of the block into country rock where they will connect with a permanent manway raise. They will be driven as small as possible, about 3 or 4 by 6 feet, leaving 5 feet of solid between them and the filled or caved stope above. Mining will be started at the end of the heading away from the manway. A 5-foot cut, 11 feet high (the standard height of top-slice floors), will be carried from the heading to the old fill or the dead line on either side. The sills of the old floor will be caught up by 8-inch round posts and head blocks. Long head blocks are used to spread the weight. Posts are stood on the ground without foot blocks. Another 5-foot cut will now be drilled and blasted. Slushers will be used to move the ore into the chutes. After each cut, or as often as an area at least 10 by 10 feet is available, floor will be laid consisting of 2-inch plank 10 feet long nailed to doubled sills of the same material. The sills are laid parallel to the direction of advance of the stope for convenience in picking them up when taking out the next floor. Two to four cuts (10 to 20 feet) will be taken out before shooting down, depending upon the weight that develops. In preparation for this, a light gob fence is built consisting of 2-inch plank nailed to the row of posts nearest the face. If the stope is being worked in two sections, it will be necessary to carry one section 30 to 40 feet ahead of the other and put up a similar fence along the dead line before shooting down the advance section. All posts except those in the gob fences will be drilled with air augers and blasted with a stick of powder each. These shots are timed so as to bring the floor down as smoothly as possible and more or less intact.

Headings are started on the floor below as soon as possible, so that when the first floor is finished mining can be started at once on the

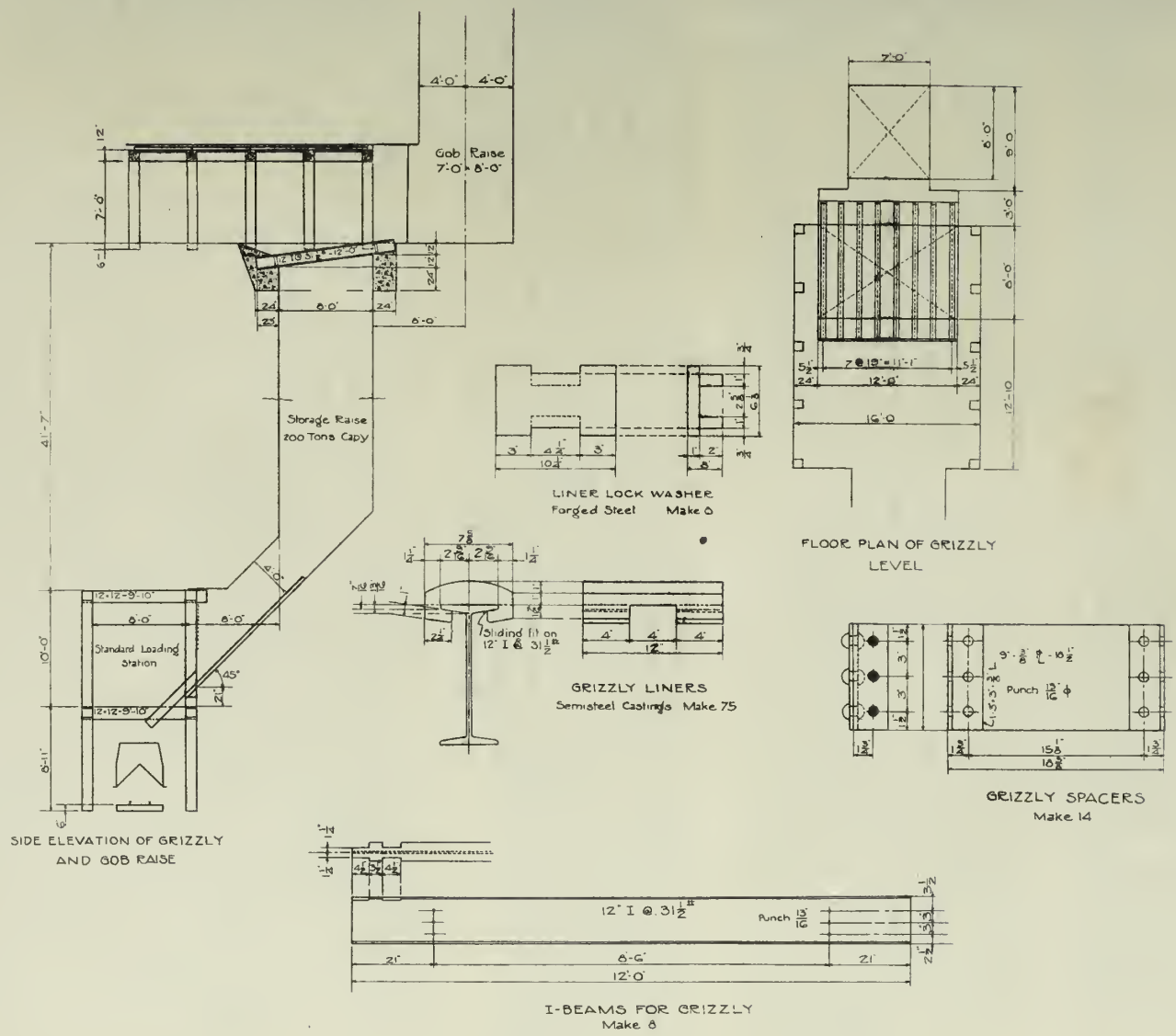


FIGURE 20.- GRIZZLY AND STORAGE RAISE

floor below. Possibly two or more floors may be worked at one time. In that case, it would be necessary to keep the lower floor at least 40 to 50 feet behind the upper to avoid interference. When two sections or two floors are being mined close to each other, it is desirable to arrange the work so that they can be shot down at the same time. Otherwise the jar and pressure adjustments caused by one coming down may cause the other to drop before it is intended. Manways in the raises will be maintained for bringing in timber and for entrance to the stope, thus giving at least two exits from each working place and aiding ventilation as well. It is believed that in these stopes both shifts can drill, blast, muck out, and timber a 5-foot cut across the stope.

Waste Glory Holes. - Waste for filling the square-set cut-and-fill stopes comes from several sources. A small amount, averaging about 200 tons per day, comes from development work. This is hoisted on the cages to the 450 level, dropped in a pocket there, and hauled by one of the waste transfer raises. Rock from any incidental surface excavation in the mine yards is used for fill. An old dump near the Kirk shaft is being used for the same purpose, loading the rocker dump waste cars with an air-operated shovel or with a drag-line scraper. All these are sources of finely-broken material which is easily handled and spread, and which, therefore, when possible is used for the last foot or two of fill in a stope.

Another source of similar material is the Kirk open-cut. A short tunnel runs from the Kirk shaft to the Colorada ore bins. Cuts were started at both portals and the broken rock was pulled through chutes in the tunnel. As the cuts advanced, chutes were put in ahead of them at 10-foot intervals and staggered on opposite sides of the drift. The rock is limestone already heavily shattered, so that it breaks finely. The maximum overburden is about 75 feet. Nine or ten men work here on day shift only, and supply about 310 tons of waste per day to the chutes. This costs about 13 cents per ton.

About half of the waste, or approximately 500 tons per day, comes from the 5-10 glory hole. This is on the hillside above the fan house at the portal of 5-10 adit. Two vertical raises, 70 feet apart, have been put up from the adit to a sublevel 50 feet above, offset there, and continued to the surface They were driven as stulled raises, the first of which was 100 feet and the second 150 feet in height above the sublevel. After holing through, they were enlarged to about 10 by 10 feet in section above the sublevel to allow large bowlders to pass through, and to 8 by 12 below the sublevel for pocket capacity. Grizzlies were installed on the sublevel (fig. 20). The raises were next belled out at the top, and glory-holing started. Ten-foot down holes are drilled with jackhammers and blasted with 40 per cent gelatin. Most of the rock is thrown into the raise by the blast, but some which is broken but not displaced is barre down by hand. The men wear safety belts when working in the pit. Work is carried on here on day shift only. The crew consists of a jigger boss and 10 to 20 men, four of whom work part of the time in the bulldozing chambers. The rock, a diorite porphyry, is hard and breaks coarsely, so that probably a quarter of it has to be broken on the grizzlies. This is done with hammers, pavement breakers or by plugging and blasting. During one month this glory hole produced waste at an estimated cost of $0.0983 per ton, delivered to the chutes in 5-10 drift.

The labor amounts to 0.229 man hours per ton (34.9 tons per man-shift) and the powder consumption is 0.12 pound per ton.

Caving. - The major part of the Capote ore body between the 700 and the 1,500 levels has been mined by various caving methods. By 1916 the ground above the 700 level had been worked out by square-setting, shrinkage, and top-slicing, and had been caved to the surface. From 1916 to 1923 considerable areas between the 700 and 1,000 levels were caved by a shrinkage stope and pillar method. Shrinkage stopes 8 feet wide separated by 12-foot pillars were laid out over the entire area to be mined. Starting from pony sets over the tramming drifts, the stopes were split to leave pillars over the drifts and then were carried up vertically for 50 or 60 feet. Sometimes they were mined to the level above, and often a long narrow stope was carried up at one or more sides of the ore body to delimit the area caved. As the stopes went up, the pillars between were weakened or entirely broken through, until the ground above began to drop. Stoping then ceased, and the caved ore was drawn out through the chutes and hand-trammed to the shaft. This method was successful in that it caved the ground. However, it required the actual drilling and breaking of a large proportion of the ore in the block and often caused loss of ore or considerable dilution due to the failure of the pillars to break down entirely between the stopes.

In 1924 a section of the ore body between the 800 and 900 levels about 75 feet wide and 150 feet long was mined by undercut caving. A narrow shrinkage stope entirely surrounding the block was carried from the fourth floor of the 900 level to the 800 level, or about 70 feet. The plug of ore thus cut off was undercut on the fourth floor by driving small untimbered drifts across it and blasting out the pillars between. The ore was drawn out through numerous chutes on the 900 level.

The area directly below this was caved in 1925 partly by undercutting with drifts on the third floor and partly by belling out and blasting down rows of raises spaced at 10 or 15 foot intervals along the tramming drifts.

On the 1,300, 1,400, and 1,500 levels the methods of caving have been alike. A fringe drift is driven around the ore body on the level, and tramming drifts at 40-foot centers are driven through from one side to the other. These drifts are timbered with 10 by 10 inch sets spaced 20 inches center to center, often with double caps. Ten-inch blocks are used for collar braces. Chutes are spaced 15 to 20 feet apart and staggered on opposite sides of the drifts. One-compartment raises are carried up from these chutes to the fourth floor, 25 feet above the rail, sometimes vertically and sometimes inclined away from the drifts to reach approximately 20-foot spacing on that floor. On the fourth floor untimbered drifts about 4 by 6 feet in section are driven parallel to the tramming drifts at 20-foot intervals and are connected by others at right angles. In very soft ground alternate subdrifts are omitted to give larger pillars, as the small pillars crush too soon. The tops of the chute raises are belled out and the pillars drilled and blasted, starting at one end of the block and retreating to the other. No boundary shrinkage stopes are necessary. However, at 30-foot vertical intervals small untimbered drifts are run around the edge of the block to create a line of weakness to which the ground will

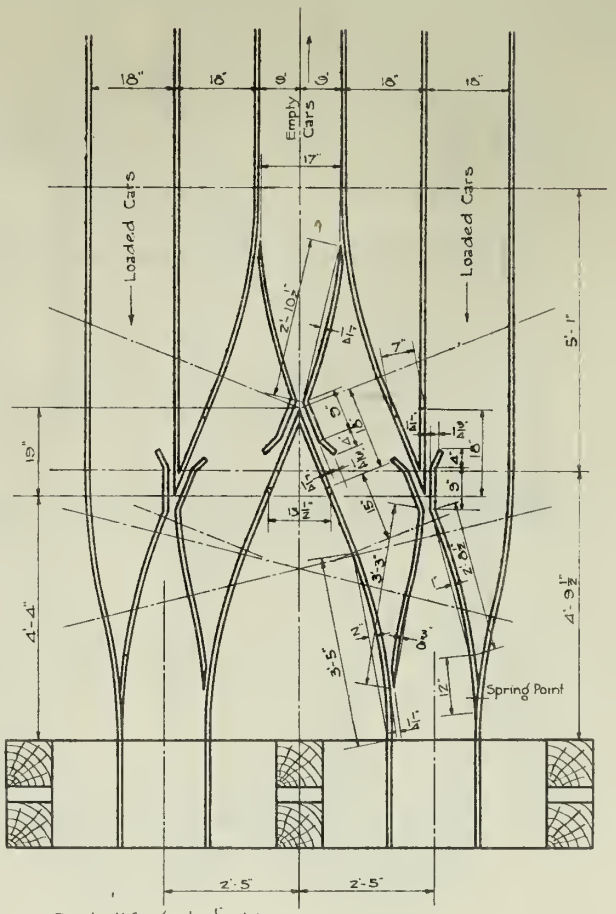


FIGURE 21.
STATION TRACKS
18" GAUGE, 10' RADIUS, 14# RAIL

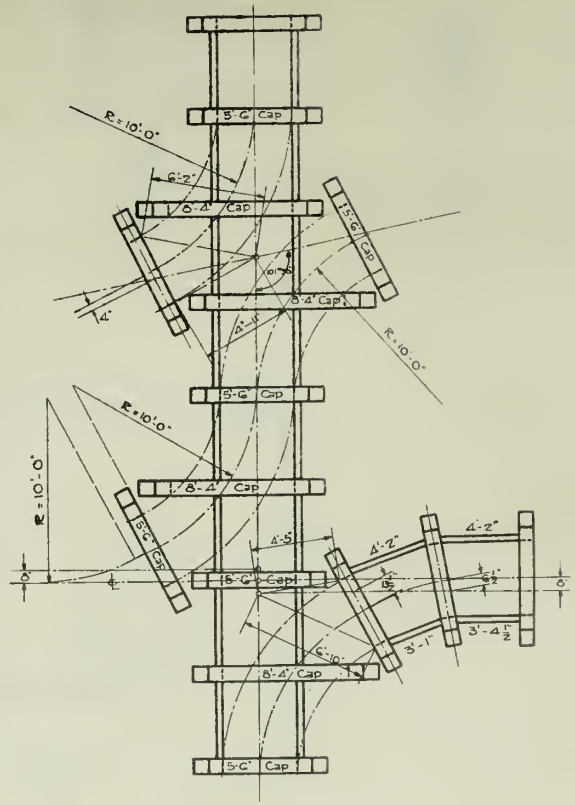


FIGURE 22.
STANDARD TWOWAY TURNOUTS
FOR ONE TON CAR
5'-6" CAPS - 10'-0" RADIUS

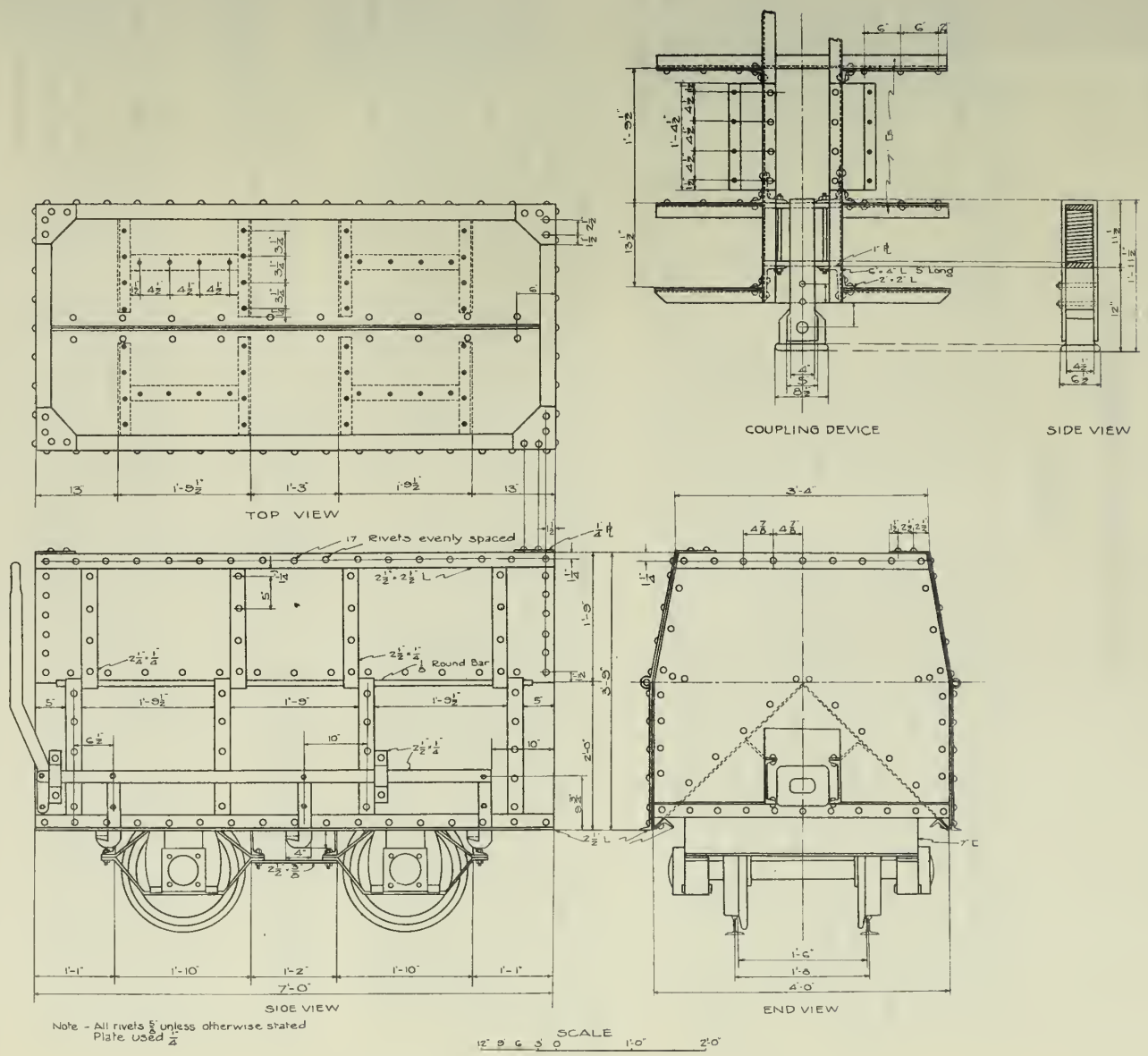


FIGURE 23. - STANDARD 4 TON MINE CAR

cave and thus reduce dilution from the waste or low-grade walls. Drawing commences from the chutes as soon as the block begins to cave and is continued as uniformly as possible until waste is reached. The ore appears to "pipe" very readily. Consequently, when a chute begins to run waste, another chute is put in close to it and a raise is driven to the broken ground. This chute will frequently produce ore for a considerable length of time. Generally the ore breaks very fine, so that there is little or no blasting in chutes. It is drawn into 1-ton cars which are hand-trammed to the shaft or made into trains for the electric locomotive.

UNDERGROUND TRANSPORTATION

Haulage

Both hand-tramming and motor-haulage are used in the producing mines. All hand-tramming is done with 20-cubic-foot (1-ton) end-dump cars weighing 1,060 pounds. These run on 18-inch gage track made with 16-pound rails laid on 4 by 6 inch ties 4 feet long and spaced 2 feet center to center. Switches for hand-tramming are of the solid type; the cars swing one way or the other as they strike the switches. Figure 21 shows the track layout on a shaft station for running cars on and off cages. Figure 22 shows standard turnouts for timbered hand-tram drifts. Almost all hand-tramming from stopes is done by contract trammers, who are paid a flat rate (20 to 25 centavos) per car from any chute on a level to the shaft station. The average performance on the 1,000 level at the Colorada, for instance, where the average tram is about 800 feet, is 25 cars per shift, though a few of the trammers will make as many as 40 under the same conditions.

Some waste haulage is now being done on the 700 level of the Colorada with two 1-1/2-ton storage-battery locomotives and 1-ton cars. These locomotives pull trains of five or six cars. Most of the waste comes from one chute. Six stopes are served by these trains, which may be operated on one, two, or three shifts, depending upon the need for waste fill in the stopes. Each locomotive has a spare battery to permit continuous operation if desired. The charging station is on the 700 level, and is equipped with two 3-kilowatt motor-generator sets and four charging racks. The usual charging period is about four hours.

These motors are now averaging about 65 tons per motor shift, but commonly handle as much as 100 tons when required to do so. The average haul is about 400 feet.

Four-ton gable-bottomed cars hauled by 4-ton trolley locomotives are in use on the 800 level of the Colorada for ore haulage. The 1,000 and 1,200 level will be similarly equipped. The cars (fig. 23) hold from 3.5 and 4.0 tons of ore and weigh 4,240 pounds empty. They are coupled by links and pins. The locomotives are the reel type of gathering locomotive, with a single pole trolley mounted on the side and one end control.

Because of the shape of the ore body on the 800 level, loop haulage could not be used without unwarranted expense. For this reason a reel locomotive was decided upon, as it eliminated the necessity of having a great many

trolley switches and also the trolley wire in front of chutes. On the 1,000 level, loop haulage with trolley locomotives is used, and chutes are therefore placed on only one side of drifts. In front of chutes a section of the wire is insulated from the line, and the power is carried by a cable.

Permanent track for these haulage levels is usually laid by special track crews. The light track used with 1-ton cars in the preliminary development work is replaced with 30-pound rails laid to 18-inch gage on 6 by 8 inch treated ties 4 feet long and spaced 2 feet center to center. Sometimes these ties are put in during the first development work so that they do not have to be replaced. The rails are bonded with soldered-terminal cable bonds pressed into holes drilled in the web of the rails. When possible grades are set at one-half per cent in favor of the load. Curves have a minimum radius of 35 feet and standard turnout timbering is used. Split switches are used for all motor-haulage turnouts.

Two locomotives on the 800 level haul trains of six to 10 cars from ore passes or stope chutes. The crews consist of a motorman and two helpers. Ordinarily the cars are spotted under the chutes and loaded without uncoupling the locomotive. Figure 8 shows the standard stope chute. The average haul to the shaft is about 800 feet. A man stationed at the shaft pocket assists in unloading the train, samples and tallies the cars, and cleans the grizzly and track. A loop drift around the shaft makes it unnecessary to uncouple the locomotive for the return trip. The cars are pulled slowly over the grizzly and dumped while in motion.

The pocket grizzlies are constructed of 50-pound rails bent down over the 12-inch I-beam girder which supports the track and is bolted at the ends to the concrete top of the pocket. The pocket on the 800 is 10 feet wide and 40 feet long at the top. It is divided into two parts by a 10-inch reinforced concrete wall. One-half is for first class, the other for second, and each has a capacity of about 190 tons of second-class ore.

The 500 level of the Colorada is now entirely equipped with 36-inch gage track. The ore cars in use there were formerly used in the Oversight and the Veta mines, and a few are still in use on the 200 level of the Veta. They are of the bottom-dump type with a level capacity of 135 cubic feet. At the Colorada they average 7.0 tons of ore per car. They are fitted with automatic 3/4-sized railroad-type couplers and spring draft rigging. The trucks have the wheels attached rigidly to the axle, and are mounted in spring-suspended half-crown bearings. The rigid wheels are the cause of excessive wear on the rails around curves, and may be the cause of occasional axle breakage. New equipment about to be applied to these cars consists of trucks with loose wheels and tapered roller bearings.

The locomotive used for ore haulage on the 500 level is a 10-ton, 2-motored, center-control locomotive. Three of these are used on the 500 level, one for ore and two for waste.

The 500-level track is of 30-pound rail laid to 36-inch gage on treated 6 by 8 by 6 foot ties spaced 2 feet center to center. The minimum

curve radius on this level is 60 feet. Grades are one-half per cent in favor of the load.

The ore comes from a pocket at the shaft. These pockets provide storage for about 200 tons of first-class ore, 650 tons of second-class ore, and 175 tons of waste. The loading chutes are similar to the stope chutes as regards timbering and control gates. They are lined, however, with 1/4-inch steel plate, and are spaced so that two chutes load one car and the next two chutes the next car, eliminating a great deal of spotting. The ore runs freely and no blasting has to be done. Normally 10 to 15 minutes are required to load a train of 12 cars. The ore bins at the surface are of timber construction and have a total capacity of about 300 tons of high-grade and 1,700 tons of mill ore.

The trains are made up of 12 cars, which are never uncoupled. The track over the surface ore bin is laid out in a loop, so that the motor remains at the front end of the train without switching. In the mine the motor has to be uncoupled and changed to the other end for each trip. This is done on a double track under the loading pocket.

The train crew consists of a motorman and one or two helpers. A man is stationed at the loading station and another at the ore bins outside. These men clean the drifts and grizzlies, assist in loading or unloading, and are also the train dispatchers. Telephones are used to obtain the right-of-way for any train or car passing through the main adit. Ore trains are given the preference.

During a recent normal period of five weeks this train hauled over 1,000 tons in one shift. The distance from shaft pocket to ore bins is 3,200 feet, 775 of which is on the surface.

During the same period referred to above, an average of 900 tons of waste per day was hauled to waste raises on the 500 level. This work is done with 4-ton rocker-dump cars and a 10-ton trolley locomotive. The cars are of 70-cubic-foot capacity and weigh 5,300 pounds. The trucks have loose wheels and are mounted in spring-suspended tapered roller bearings. The cars are also equipped with 3/4-sized, automatic, railroad-type couplers and spring draft rigging.

The waste train coming from the Kirk glory hole travels in part over the same line as the ore train; the total distance, however, is about 4,200 feet. Waste from the main glory hole over the 5-10 adit is trammed only about 2,000 feet. At the Kirk the cars are loaded one at a time under ordinary stope chutes. At the 5-10 adit there are two loading stations, one under each bull-dozing chamber. Each has two chutes similar to those under the ore pocket at the shaft.

A waste train crew consists of a motorman and two helpers.

The cars are estimated to hold 3-1/2 tons of the Kirk waste. The train pulling from there is usually made up of six to 10 cars and works three 8-hour shifts per day, averaging 50 cars or 175 tons per shift, with sometimes as much as 75 cars per shift. In addition, this train handles 10 to 20 truck-

loads of timber and other supplies from the Colorada yards to the shaft. The trucks for this service are all-steel platform trucks with decks 4 feet 6 inches wide, 12 feet 0 inches long, and 2 feet 8 inches above the rail. They are equipped with tapered roller bearings.

The rocker-dump cars hold 4 tons of the coarser waste from the 5-10 chutes. The train hauling from there works only on the regular day and night shifts. With a crew of one motorman and two helpers, it hauls an average of 60 cars or 240 tons per shift, and often as many as 100 cars. The train is made up of 10 cars.

Sometimes fine material builds up in the bottom of the rocker-dump cars and also in the small 20-cubic-foot cars. Compressed-air clay diggers are used to clean out this material, and cause much less wear and tear on the car bottoms than results from the use of 8-pound hammers for the same purpose.

Hoisting

When the development of the mine is completed, all ore will be hoisted in skips from the 800, 1,000, and 1,200 haulage levels. Ore from stopes on other levels will be dropped through ore passes to loading chutes on those levels. At present, however, a large proportion is being trammed in 1-ton cars to the shaft and caged to the 450 level.

Two compartments of the shaft are devoted chiefly to this type of hoisting. The cages are three-decked. The upper deck is equipped with a roof and safety dogs, and weighs 4,100 pounds. The two lower decks are alike; each weighs 2,200 pounds, thus making the total weight of the cage 8,500 pounds. The cage holds 27 men, nine to a deck, or three 1-ton cars. It is operated by a double-drum, 72 by 72 inch, electric hoist on the surface. The drums are grooved, holding 2,500 feet of cable in something over two laps. The cable is standard lay 6 by 19 and 1-1/8 inches in diameter. The hoist is driven through herringbone gears by a 400-hp., 2,200-volt, induction motor with a speed of 595 r.p.m. under full load. The hoisting speed is 1,000 feet per minute. The hoist is equipped with a safety device which prevents overspeed or overwinding, and applies the brakes in case of current failure. Another safety measure is a mechanical interlocking of the control levers in such a manner that the clutches can not be thrown out while the brakes are off.

This hoist is now lifting about 475 cars of ore and waste per day, running on three eight-hour shifts, in addition to lowering and hoisting the shift and handling some timber and other supplies.

One-ton cars are hoisted to the 450 level, trammed about 50 feet across the station, and dumped into the pocket for the motor haulage. Regular lever-operated landing chairs were installed in the shaft, but these have been abandoned in favor of a device consisting chiefly of three heavy links of chain. The links are attached by an eye-bolt and nut to a heavy bracket fastened permanently to a shaft post near the edge of the shaft and about 3 feet above the station floor. The cage is fitted with a heavy solid hook near the floor on one corner post. As the cage is lowered slowly to the station level,

the cager swings the chain toward the shaft so that the lower link engages the hook on the cage, which then comes to a stop at exactly the right position for unloading the car. When the cage is hoisted, the chain disengages itself automatically and swings into the clear. When hoisting from that level is finished for the time, the cager disconnects the chain and takes it to the next level. This is simpler and just as quick to operate as the old-fashioned chairs, and eliminates repairs and possible shaft accidents.

The hoisting crew for each shift consists of a hoist engineer and an oiler and two cage tenders. In addition, four men work on the 450 station tramming to the pocket.

Skip-hoisting is done with a hoist identical with that operating the large cages. Power and hoisting speed are the same.

The skip-loading devices on the 800 and 100 levels consist of two measuring hoppers and two storage hoppers. Air-operated, down-dropping, vertical steel gates allow ore to run into the measuring hoppers, one below the mill ore chute and one below the high-grade chute. Under the hoppers is a tilting slide which can be thrown to divert the ore from one side to the other. Below the slide are two storage hoppers, one for either compartment. The ore remains in these until the skips are spotted, and then other vertical-sliding air-controlled gates are raised, allowing the ore to drop into the skips. These lower hoppers were put in partly to prevent the spillage and wear on the skips which would be caused by dropping the ore the considerable distance from the upper chutes and over the slide to the skip, and partly because loading can be done more promptly and rapidly. Only about five seconds is needed to load a skip.

The skip is a standard for all the shafts in the district. It is 4 feet by 2 feet 10 inches inside, 6 feet 4 inches deep at the front, and 7 feet at the back. Its level-full capacity is 76 cubic feet, but only 60 cubic feet is used in practice to eliminate any spillage due to slight overloading at the pocket or to the shock which the skip receives upon entering the dumping cradle. The skip is equipped with the usual eccentric, toothed-cam, safety dogs. Its weight is 4,660 pounds, and the usual load is 3.5 tons of ore. The skip dumps about 130 feet above the 500 level into an inclined steel chute 2 feet 6 inches by 3 feet 6 inches in section and made of 1/4-inch plate. This chute drops on a 45° pitch to a point directly over the motor-haulage pockets. There the ore is thrown to either the first or second class pocket by means of a diverting gate. The floor of the chute is lined with old rail, and just above the diverting gate the ore hits a swinging curtain made of old rail, which checks its speed.

The crew on the skip hoist on each of three shifts consists of a hoist engineer, an oiler, and two skip tenders. The average hoisting duty now is about 350 skips per shift, though often as many as 450 have been hoisted in one shift.

A double-drum, double-geared electric hoist is used for the dinkey cage in the Colorada shaft, the cages in the ventilation shaft, and several

other installations in the camp. The drums are 42 inches in diameter and 30 inches wide and are grooved for 3/4-inch rope. The motor is 112 horsepower and the hoisting speed is 495 feet per minute. As manufactured it was equipped with band brakes, but in several cases these have been replaced by post brakes. Brake shoes are cut from 1-inch boiler plate and fitted with angle irons to which the brake band is attached. Asbestos brake blocks are giving excellent service in place of the basswood formerly used. This hoist is very satisfactory to depths of 500 to 600 feet and is often used for considerably greater depths.

COMPRESSED-AIR SUPPLY

The present supply of compressed air for the camp comes from three sources. A large steam-driven compressor at the smelter at Ronquillo and electric-driven machines at the Sierra de Cobre and Colorada are all connected to the same lines.

The Ronquillo compressor, with a capacity of approximately 4,500 cubic feet per minute, is connected to the mines by a 10 and 12 inch pipe line as far as the Veta Grande, and thence to the Colorada by a 6-inch line. The latter section is known to be the cause of a serious pressure drop and is to be replaced by larger pipe.

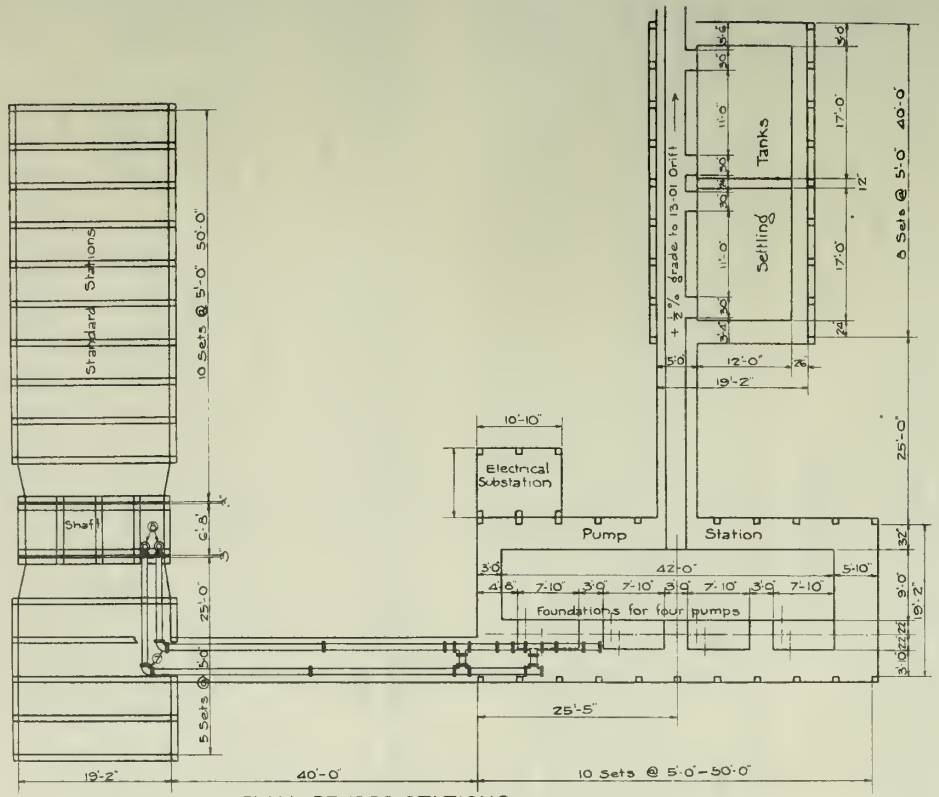
On the surface at the Colorada shaft is a twin angle-compound direct-connected compressor, 22 by 13 by 14 inches, driven at 225 r.p.m. by a 2,200-volt, 450-hp., synchronous motor. Its capacity is 2,200 cubic feet per minute.

At the old Sierra de Cobre are two horizontal duplex compressors. One is 17-1/2 by 22 by 21 inches driven at 180 r.p.m. by a direct-connected, 422-hp., 2,200-volt, synchronous motor. Its capacity is 2,874 cubic feet per minute. The other machine is of the same make and type but with cylinders 16-1/4 by 28-1/2 by 24 inches, driven at 180 r.p.m. by a 350-hp. induction motor. It has a capacity of 2,430 cubic feet per minute.

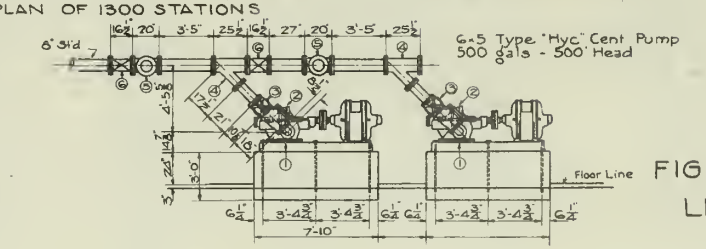
The Colorada alone is now using approximately 6,000 cubic feet per minute for about 75 drills and 10 scraper hoists in actual daily operation. Air is delivered to these machines at an average pressure of 85 pounds per square inch.

PUMPING

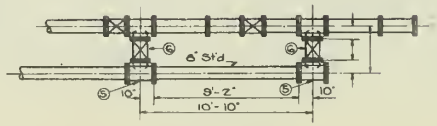
No large amounts of water are contended with in any of the mines. The Capote is now pumping 1,100,000 to 1,200,000 gallons per day, or 750 to 850 gallons per minute. Practically all of this is collected on the 1,600 level and pumped to the 1,200 level, from which it is relayed to the surface. In the 1,600 pump station there are three two-stage horizontal centrifugals, 6 by 5 inches, driven at 1,760 r.p.m. by 150-hp. motors. Their rated capacity at 500-foot head is 500 gallons per minute. The actual head is about 410 feet. On the 1,200 level there are three 5-1/2 by 12 inch triplex pumps with capacities of 370 gallons per minute at 60 r.p.m. They are driven by 125-hp. motors. A fourth pump on this level is a 6-1/2 by 14-inch triplex pump driven by a 150-hp. motor. This unit has a capacity of 300 gallons per minute at 52 r.p.m. The head on the 1,200 level pumps is 1,100 feet.



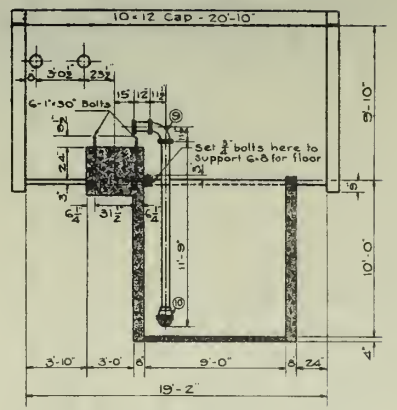
PLAN OF 1300 STATIONS



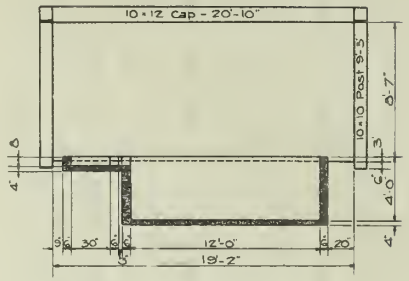
1 5"x6" Elbow Reducer
2 6" Check Valve
3 6" Gate Valve
4 8"x8"x6" Lateral
6 8" Gate valve
5 8"x8"x8" Tee
7 8" L R Elbow
8 8" Sq Base L R Elbow
9 6" L R Elbow
10 6" Foot Valve
All fittings flanged C.I



PIPING DETAILS



SECTION THROUGH PUMP STATION



SECTION THROUGH SETTLING TANKS

FIGURE 24. - PUMP STATION ON 1300 LEVEL, LA COLORADA MINE

SCALE FEET
0 2 4 6 8 10 12 14 16



In the Colorada mine all the water is pumped from the 1,300 level to the 900 and thence to the 500 levels, where it flows out the main adit to the surface. Figure 24 shows the 1,300-level pump station. Only two units have been installed, but the foundation is ready for a third should it be needed. These pumps are 6 by 5 inch, horizontal centrifugals identical with those on the Capote 1,600 level. Two of the same pumps are installed on the 900 level. The actual head on both the 1,300 and 900 level pumps is 500 feet. The flow of water in the mine has increased steadily during 1929 from 56,000 gallons per day in January to 120,000 (83-1/3 gallons per minute) per day in September partly because of flows of water encountered in a long prospect drift being driven on the 1,300 level. The present flow is handled by one centrifugal pump on the 900 and one on the 1,300 level, which run an average of 4 hours per day. It is the experience of the district that very large temporary flows of water may be struck when least expected, and the mines are therefore equipped to handle much more than the average flow would seem to require.

The Veta Grande is the only mine now operating in which any relation can be traced between the monthly rainfall and the amount of water pumped. Two pumps are handling about 94,000 gallons per day (65 gallons per minute). One is a 5-1/2 by 9 inch triplex, and the other a 6 by 9 inch pump. Both are driven by 25-hp. motors and have capacities of 175 gallons per minute. The average pumping time is about 10 hours per day for one pump.

VENTILATION

Ventilation of the Colorada mine is provided by a reversible 70,000-cubic-foot fan at the portal of a 500-level adit. The air intakes through the main haulage drift on the 500 level and also down the Colorada shaft, the two streams joining at the shaft station. Air from the shaft passes across the several levels by way of the haulage and development drifts and upcasts in an air raise on the opposite side of the ore body. Regulating doors between the working areas and the air raise control the amount of air coming off on each level and create a pressure difference that causes circulation upward through the stopes from level to level.

Figure 25 shows the air-operated ventilation door used on motor-haulage levels. The control levers are placed at the side of the drift, more than a train's length away from the door, and are connected to the valve of the air cylinder by 1/4-inch wire rope.

The three thicknesses of board make a strong and tight door, but must be laid symmetrically. A diagonal layer in the center was tried, with the idea of making a stronger door, but it was found that this caused decided warping. The door closes into a recess in the frame to reduce leakage around the edges. When a water ditch is required, it is concreted under the door frame. A small canvas curtain with a strip of wood at the lower edge to weight it down is then hung from the door to the bottom of the ditch. When water is flowing the wood floats but still holds the canvas to the surface of the water, thus preventing air leakage.

The clear area of the Colorada shaft is 100 square feet, exclusive of the manway compartment. The 500-level haulage adit, which is a main

intake, is at least 8 by 8 feet in section, and is generally larger. Drifts on other levels are at least 6 by 8 feet in the clear, whether timbered or untimbered. The air raise has three compartments 5-1/2 by 6 feet inside, two of which are used as hoisting compartments; the third is a manway with landings at every 16 feet. A tunnel 8 by 8 feet in the clear, untimbered, leads from the top of the air raise to the fan. As shown in Figure 4, the ventilation system is not complete to the 1,300 level; the 1,000 level is the lowest now connected to the air raise. Development raises serve to ventilate the lower levels until the air raise is finished.

ACCIDENT PREVENTION, FIRST-AID, AND MINE-RESCUE TRAINING

Safety First work at Cananea is in charge of a Safety-First director. After daily trips through the mines and plant, he makes written reports to the mine superintendent on the condition of each working place as regards safety. The existence of fire hazards and the condition of manways, ore passes, grizzlies, shaft stations, drift timber, powder magazines, and machinery are among the items noted in these reports. Falls of rock are the greatest accident hazard at present, and emphasis is laid upon barring down before starting work. All machinery is fitted with safety guards. Small hinged grizzlies cover the tops of all timber slides. Men working where serious falls might occur are required to wear safety belts. Goggles are worn when breaking rock with hammers or moils. Hard-boiled hats are compulsory for all persons going underground. As an added incentive, jigger bosses whose crews work 1,000 man-shifts without a time-lost accident are paid a bonus.

Another phase of the safety director's work is the training of men in first aid. Special attention is paid to the stopping of arterial bleeding and the use of artificial respiration. The course consists of eight or 10 lessons. The classes are held outside working hours, but with extra compensation for the men who attend. All bosses in the mine carry first-aid kits and are competent to use them, and several hundred men have likewise been given the first-aid course.

The ores being mined at present are generally low in sulphide, so that no fire hazard exists from heating and burning of the ore. However, considerable attention is paid to the prevention and combating of possible fires from other sources. All of the permanent timbering is now done with treated timber, which is fire-resisting. The shaft was sunk before the treating plant was in operation, however, and only the skip compartments, added at a later date, have treated timber. The shaft is therefore protected in other ways. A line of sprays is installed in it at 25-foot vertical intervals. These are turned on at intervals to keep the timber permanently wet. For flooding the shaft in case of fire, one water line at the collar can deliver 300 gallons per minute and another at the 500 level can contribute the same amount. These are fed from two storage tanks on the surface. Above each station a 10-inch ring of concrete has been poured around the shaft and extends from the rock to the outside of one set. The concrete will prevent a fire from working upward back of the wet lagging and will also bring in water which may be running down the walls. Fire hydrants and hose, as well as chemical extinguishers, are installed on each shaft station. These are tested periodically.

The shaft can be completely cut off from the mine workings by means of steel fire doors near it on every level. These can all be closed at once by opening an air valve at the surface or at the 500, 800, or 1,000 level stations. In case of fire in the shaft, the doors would be closed and the fan would be reversed, thus putting the shaft under pressure and keeping the smoke and gas out of the mine. Men could then escape by way of the air raise and ventilation tunnel on the 500 level.

Junction boxes and electric switches on the shaft stations are enclosed in concrete boxes. Inflammable waste material is not allowed to accumulate in the mine. Each electric locomotive carries a small extinguisher. If fire occurs, ethyl mercaptan can be put in the air lines to warn men to come out of the mine. Finally, about 40 men have been given training in mine-rescue work and the use of oxygen breathing apparatus.

WAGE, CONTRACT, AND BONUS SYSTEMS

Between 50 and 60 per cent of the whole underground force is employed on a contract basis. Practically all raising and the larger part of drifting are done on contract. Some timbering of drifts, practically all tramming, and a large part of the stoping are contracted for. All men thus employed are contractors in relation to the company and share equally in the contract payment.

Development work is contracted at so much per foot of advance, timbering per set, tramming per car or per ton, and stoping per cubic meter. The company furnishes everything except labor and explosives. Contracts are generally made out for no specified amount of work and are paid off and renewed each half-month. Even when contracts are given for definite amounts of work, requiring more than two weeks to finish, full payment is made after each period for all work completed, and nothing is held back pending completion of the work. Informal bonuses are often paid for difficult work, such as completing the last of a long raise.

ADMINISTRATIVE ORGANIZATION

Under the general mine foreman are four or five mine foremen, one at each operating mine. On night shift each mine is in charge of an assistant or night foreman. Next in authority are the American shift bosses, up to five or six on a shift at each mine. These men are assigned to definite parts of the mines and are in charge of all operations within their territories, without regard to kind of work. They are each assisted by from one to five Mexican jigger bosses who are directly in charge of the men. Each jigger boss likewise covers a definite territory, such as two or three stopes, or a number of development headings. Their crews range from 10 to 40 men, averaging about 25.

Table 4. - Summary of costs, Colorada mine, first six months of 1929 1/

Tons of ore produced during the period: 262,278

	Tons
Rill cut and fill	104,057
Horizontal cut and fill	37,949
Square-set	32,220
Shrinkage	36,354
Drifting	51,698
	262,278

	Underground costs per ton of ore produced							
	Labor	Super-vision	Comp. air, drills, and steel	Power	Explo-sives	Timber	Other sup-plies	Total
Development:								
Drifting and raising; In ore and in rock	.2260	.0396	.0540	.0151	.0640	.0591	.1032	.5610
Mining	.3748	.1316	.1808	.0071	.2146	.1979	.0966	1.2034
Transportation	.1126	-	-	.0250	-	-	.0288	.1664
General underground	.2115	-	-	.0120	-	-	.0541	.2776
Surface expense	.1767	-	-	.0107	-	-	.0452	.2326
Totals	1.1016	.1712	.2348	.0699	.2786	.2570	.3279	2.4410

1/ These costs exclude mine departmental expense, which includes superintendence, engineering, sampling, watchmen, time office, etc.

Table 5. - Summary of costs, Capote mine, first six months of 1929 1/

Tons of ore hoisted during the period: 142,168

Mining method: Undercut caving, top slice, and cut and fill

	Underground costs per ton of ore hoisted							
	Labor	Super-vision	Comp. air, drills, and steel	Power	Explo-sives	Timber	Other sup-plies	Total
Development:								
Drifting and raising: In ore and in rock	.0196	-	.0018	.0111	.0094	.0068	.0100	.0577
Mining	.2923	.1188	.1023	.0200	.0995	.1725	.1187	.8941
Transportation	.0187	-	.0001	.1416	-	.0007	.0404	.2015
General underground	.0316	-	-	.0100	-	-	.0412	.0828
Surface expense	.1925	.0135	.0071	.0110	-	.0017	.0163	.2421
Totals	.5247	.1323	.1113	.1937	.1079	.1817	.2266	1.4782

1/ These costs exclude mine departmental expense, which includes superintendence, engineering, sampling, watchmen, time office, etc.

Table 6. - Summary of costs, Veta Grande mine, first six months of 1929 1/

Tons of ore hoisted during the period: 42,565

Mining method: Square set, top slice, and cut and fill

	Underground costs per ton of ore hoisted							
	Labor	Super-vision	Com. air, drills, and steel	Power	Explo-sives	Timber	Other sup-plies	Total
Development:								
Drifting and raising:								
In ore and in rock	.1587	.0194	.0230	.0097	.0258	.0648	.0212	.3226
Mining	.6377	.0941	.1323	.0283	.1498	.3524	.1341	1.5287
Transportation	.0741	-	.0012	.0095	-	.0111	.0119	.1078
General underground	.0468	.0182	.0001	.0177	-	-	.0393	.1221
Surface expense	.0900	-	-	.0010	-	.0125	.0118	.1153
Totals	1.0075	.1317	.1566	.0662	.1756	.4408	.2183	2.1965

1/ These costs exclude mine departmental expense, which includes superintendence, engineering, sampling, watchmen, time office, etc.

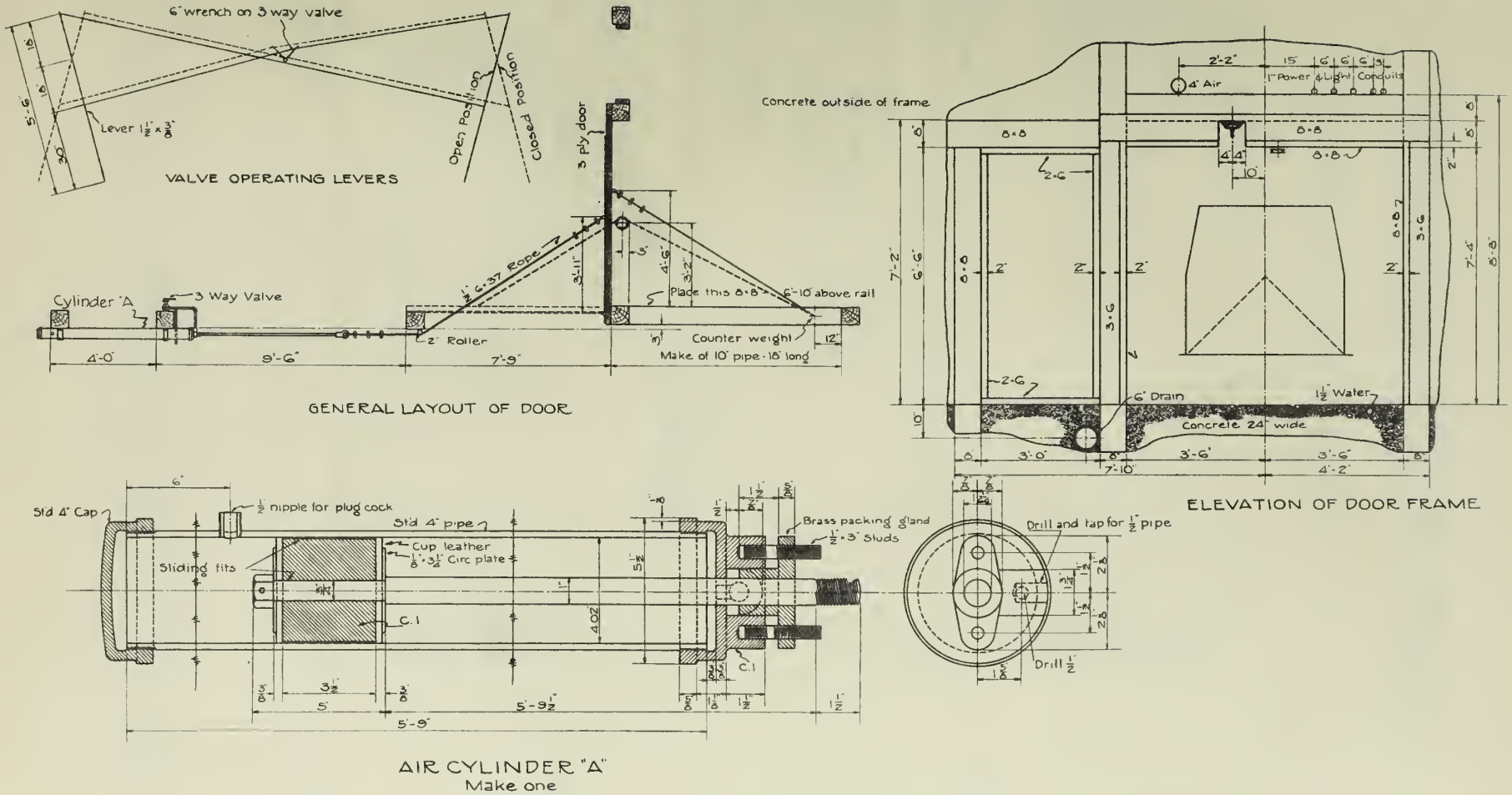


FIGURE 25. – VENTILATION DOOR FOR MOTOR HAULAGE DRIFTS

Table 7. - Summary of costs in units of labor, power, and supplies, Colorada mine, first half of 1929

Tons of ore hoisted: 262,278

A. Labor (man hours per ton):

Breaking	.876	
Timbering	.799	
Mucking	.990	
Haulage and hoisting	.494	
Supervision	.297	
General	.163	
Total underground		3.619
Tons per man-shift underground		2.21
Surface labor, directly chargeable	.531	
distributed charge	.209	
Total surface		.740
Tons per man-shift on surface		10.81
Tons per man-shift, total labor		1.835
Labor, per cent of total cost		52.10

B. Power and Supplies:

Explosives (lbs. per ton, 85% of 40% gelatin 15% of 60% gelatin)		1.37
Timber, bd. ft. per ton		7.19
Total electric power, kw.h. per ton		11.291
Motor haulage	0.986	
Air compression (part of air only)	6.045	
Hoisting	3.373	
Pumping	0.454	
Ventilation	0.195	
General underground	0.238	
Other supplies in per cent of total supplies and power		30.01
Supplies and power, per cent of total cost		42.16

Note: 5.74 per cent of total cost is for several distributed items such as shop overhead, narrow-gage railroad, and sampling and assaying, in which labor and supplies could not be satisfactorily separated.

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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

METHODS AND COSTS OF STRIPPING AND MINING AT THE UNITED VERDE OPEN-PIT MINE, JEROME, ARIZONA.

BY

E. M. J. ALENIUS

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METHODS AND COSTS OF STRIPPING AND MINING AT THE UNITED VERDE OPEN-PIT MINE, JEROME, ARIZ.[1]

By E. M. J. Alenius[2]

INTRODUCTION

This paper describing the mining practices at the United Verde open-pit mine at Jerome, Ariz., is one of a series being prepared by the United States Bureau of Mines on mining practices, methods, and costs in the various mining districts in the United States.

The purpose of this paper is to present the methods used in developing and operating the mine, and to discuss the problems encountered.

The United Verde open-pit mine is situated in Bitter Creek Gulch on the east slope of the Black Hills, directly west of Jerome, in the central part of Arizona. The elevation of the surface is 5,490 feet above mean sea level. West of the mine the slope of the hills continues to the flat-topped Mingus Mountain which reaches an altitude of 7,720 feet; to the east the hills slope towards the Verde River which has an elevation of 3,350 feet at Clarkdale.

ACKNOWLEDGMENT

Mr. J. C. Perkins, superintendent of the steam-shovel department of the United Verde Copper Co., rendered valuable assistance in the preparation of this paper by supplying information relative to the open-pit operations.

HISTORY

Although the United Verde mine has been operated since 1876, open-pit mining with power shovels was not begun until the fall of 1918. Shovel operations had been preceded by glory-hole mining on the south end of the ore body. The upper portion of the ore body was originally mined by underground methods. However, in 1893 a fire occurred in the upper workings, which necessitated the abandonment of several high grade stopes and increased the cost of mining in the adjacent areas. Later subsidence took place, seriously affecting the smelter and

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6248."

2 One of the consulting engineers, U. S. Bureau of Mines.

surface plant which had been constructed on top of the ore body (fig. 1). For 25 years efforts were made to mine the ore in the fire areas and to reduce the fire, but without success. At the end of this time Mr. R. E. Tally, general manager of the United Verde Copper Co., devised an open-cut method for mining the ore in the fire zone.

In 1920 stripping was advanced to permit the removal of a small amount of schist ore. During 1921 and for three months in 1922 operations were suspended on account of the low price of copper. Since 1922 mining has rapidly advanced, and the major stripping operations were completed in October, 1927.

Yardage removed up to January 1, 1929

Year	Stripping, cubic yards	Ore, cubic yards	Total, cubic yards
1919	374,260		374,260
1930	1,026,382	5,760	1,032,142
1922	486,260	132,481	618,741
1923	882,524	361,912	1,244,436
1924	1,359,645	142,740	1,502,385
1925	1,653,009	165,084	1,818,093
1926	1,323,991	210,479	1,534,470
1927	570,869	230,007	800,876
1928	322,538	276,055	598,593
Total, Jan. 1,1929	7,999,478	1,524,518	9,523,996

GEOLOGY

The general geological features at Jerome are well exposed on the rough slopes of the Black Hills. The summit of the Black Hills is covered by Tertiary basalt resting upon horizontal Paleozoic limestone and some sandstone. Next below lie the pre-Cambrian rocks, which are very evident at Jerome. These rocks consist chiefly of the so-called greenstone complex. Intruded in the greenstone are masses of rhyolite porphyry and diorite. The United Verde ore body lies in massive pyrite which is between adjacent masses of the rhyolite porphyry and diorite. The pyrite is a replacement deposit of schist, formed in the overlying concave trough of the United Verde diorite.

The Verde fault cuts through Jerome below the exposed pre-Cambrian rocks in a general northwesterly direction. The fault has a throw of about 1,700 feet. North and east of Jerome, on the other side of the fault and at a lower elevation than the pre-Cambrian and Paleozoic beds, the Tertiary basalt is again apparent, covering the successive rocks. Towards the Verde River these downthrown rocks are more or less covered with Quaternary gravels.

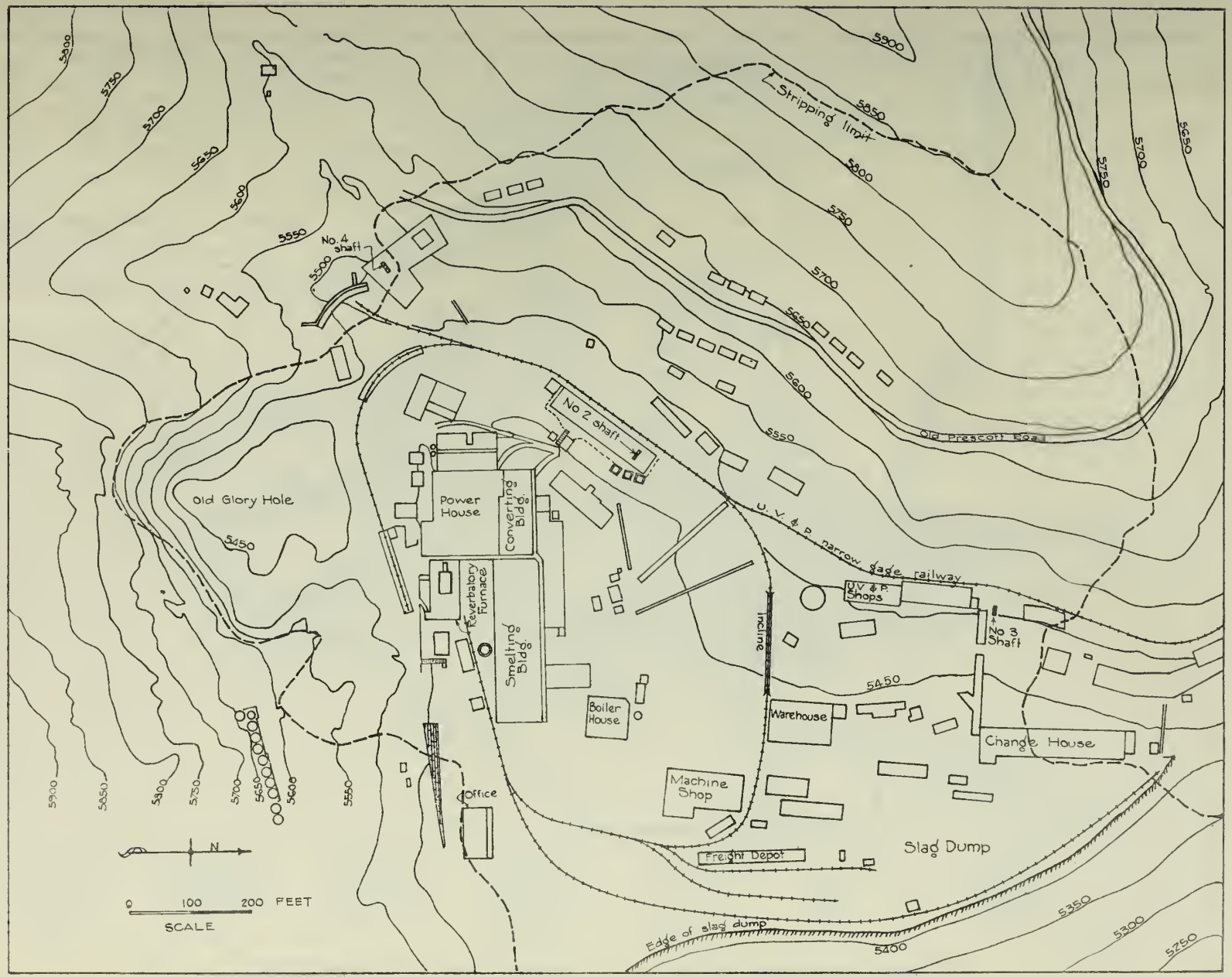


FIGURE 1.- PLAN OF UNITED VERDE SURFACE PLANT AND SMELTER, BEFORE STEAM SHOVEL OPERATIONS

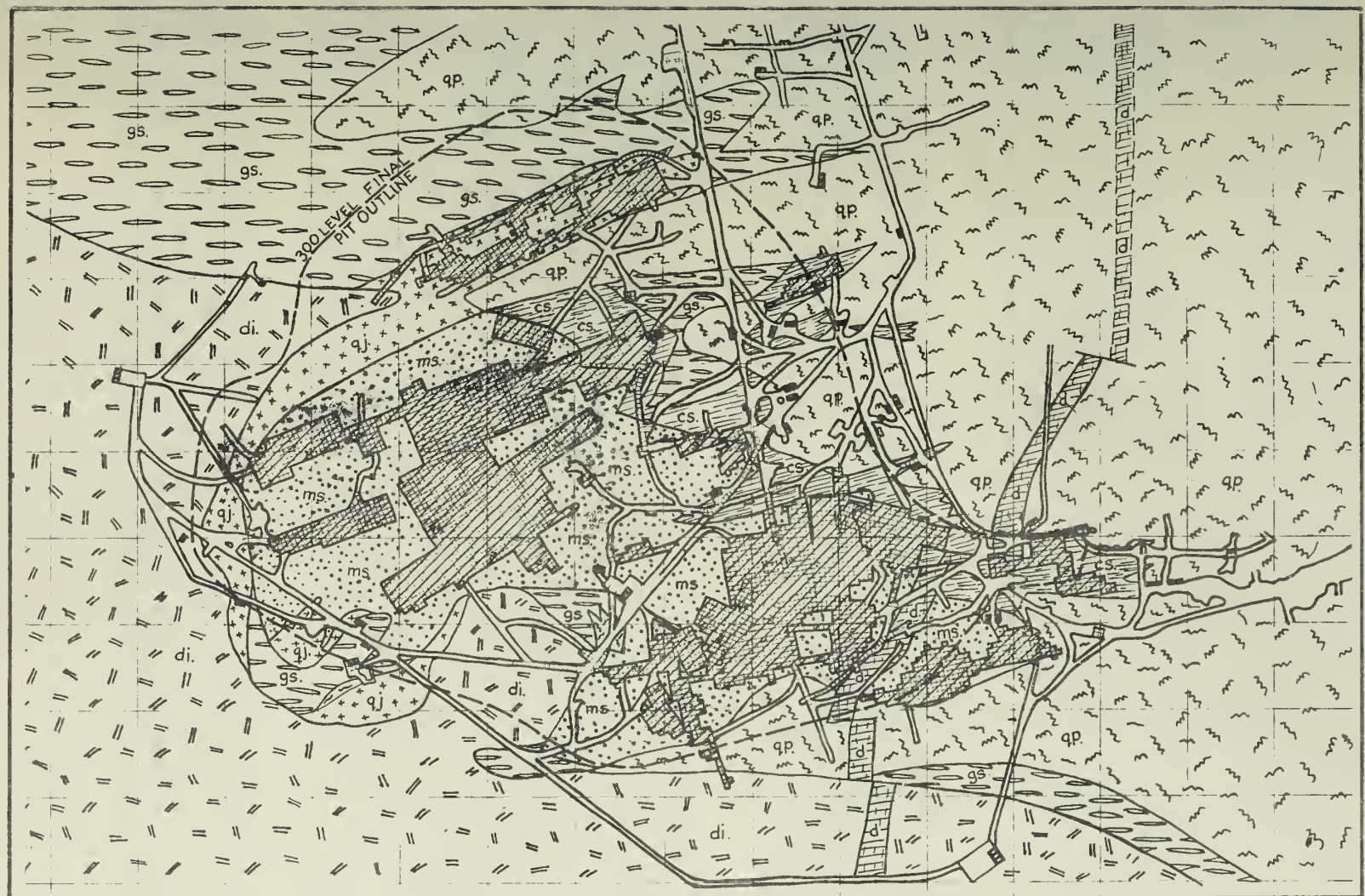


FIGURE 2.- PLAN OF 300 LEVEL SHOWING GEOLOGY AND OLD UNDERGROUND WORKINGS

ORE DEPOSIT

The massive-sulphide ore body, consisting mostly of pyrite with associated copper and zinc sulphide, is a pipe-like mass resting on the quartz-rhyolite porphyry footwall and supporting a dense barren diorite hanging wall. The zone containing copper ore is approximately 1,000 feet by 800 feet in plan on the 300-foot level (fig. 2). It dips steeply to the northwest and has been opened to the depth of 3,000 feet. The upper portion of the sulphide ore body has undergone considerable oxidation, which extends roughly to the 160-foot level. In the oxidized zone the ore consists chiefly of copper oxides and quartz carrying gold and silver. Below the zone of oxidation the massive sulphides are enriched with the secondary copper minerals, chiefly chalcocite. On the footwall side of the ore body the massive sulphides are more or less separated from the porphyry by sulphide-bearing schist shoots extending into the quartz porphyry. The massive sulphide is low in silica adjacent to the schist, and increases in silica as the diorite is approached. It is generally separated from the diorite by barren quartz and jasper.

The ore material is in general divided into three classes--direct smelting, concentrating, and converter. The direct-smelting ore consists chiefly of the massive sulphides. Clean mineralized schist and siliceous sulphide are sent to the concentrator, whereas mineralized material containing 60 per cent of silica or better is used for converter flux. Some material obtained in stripping operations which is too low grade for milling or smelting is piled for leaching.

PRELIMINARY PREPARATION AND PLANS

Before shovel operations could be started it was necessary to select new sites for the smelter and mine surface plant. A new smelter was constructed west of the Verde River at Clarkdale, 5 miles below Jerome on the Atchison, Topeka & Santa Fe Railway, and began operating in 1915. Shortly afterwards the present mine plant was built at the elevation of the 500-foot level, about 1/2-mile north of the main workings, and an adit was run on that level into the main ore body for the transportation of men and material. New shafts with underground hoist rooms and headframes also had to be completed before the entire upper portion of the ore body became completely available for the pit operations. For the removal of ore from the underground mine the Hopewell Tunnel was run on the 1,000-foot mine level to cut the slope of the hills about $1\frac{1}{4}$ miles north of the mine. The Verde Tunnel & Smelter Railroad was constructed to transport ore from the portal of the tunnel to the smelter at Clarkdale and later was extended to Jerome so that the heavy equipment necessary for the pit operations could be brought up to Jerome.

The first plan of operations called for exposing the entire ore body on the 300-foot level and extending the walls above that level into the porphyry and diorite waste on a one-half to 1° slope. Below the 300-foot level a part of the ore body was to be mined to the 400-foot level by glory-holing. Accordingly, operations were started at the intersection of the 300 level with the slope, at the point closest to the ore body. At the same time switchbacks on the side of

the diorite hill were constructed so that the upper part of the bank might be removed on a series of benches approximately 50 feet apart. Work was conducted on two of the upper benches around the level of the old smelter and mine plant. The upper bench would be at an elevation of 5,850 feet or about 640 feet above the 300-foot level. Mining was carried on according to this plan until the shutdown in 1921.

From geological data obtained in the underground workings it was estimated that approximately 1,900,000 cubic yards or 4,000,000 tons of ore remained above the 400-foot level. In order to make this available the original plan required the removal of 14,000,000 cubic yards of stripping. During the suspension of operations the original plans were revised. Instead of cutting an approach on the 300 level, the cut was to be made on the 160 level, and glory-hole operations were to be carried from the 160 level to the 500 level. This change of plan was deemed advisable on account of the fact that the 160 level is roughly the dividing level between the zones of oxidation and secondary enrichment, and also because the change would decrease the total yardage to be stripped. The yardage would be further decreased by steepening the slope of the pit wall. At this time it was believed that the material of the final bank would stand on a one-half to 1° slope. Berms were planned for the final wall, two to be 20 feet wide and one 30 feet wide on the 160 level. These berms were to serve for drainage and for keeping the banks trimmed. At the resumption of operations in 1922 it was estimated that 7,000,000 cubic yards of stripping remained to be removed to make available 2,400,000 cubic yards of ore.

Thus two distinct methods of mining were employed at the United Verde open pit. The first operation extended above the 160 level and consisted chiefly of stripping and removing the oxidized ore by steam shovels working on benches. The second extended between the 160 level and 500 level and was confined chiefly within the ore zone. The original revised plan was to mine the ore by means of mill holes, but conditions arising later made necessary a revision of the plans. However, the two methods of operation are entirely distinct, and in continuing this paper, methods and costs for the upper-pit or major stripping operations will be discussed in detail separately from the lower-pit operations. Yardages involved in the two divisions are as follows:

Upper and lower pit yardage

Major stripping operations:	Removed to Jan. 1, 1929	Balance remaining	Total yardage
Ore	803,354	- -	803,354
Waste	7,274,097	- -	7,274,097
Total	8,077,451	- -	8,077,451
Lower-pit operations:			
Ore	721,164	2,121,822	2,842,986
Waste	725,381	796,357	1,521,738
Total	1,446,545	2,918,179	4,364,724
Grand total	9,523,996	2,918,179	12,442,175

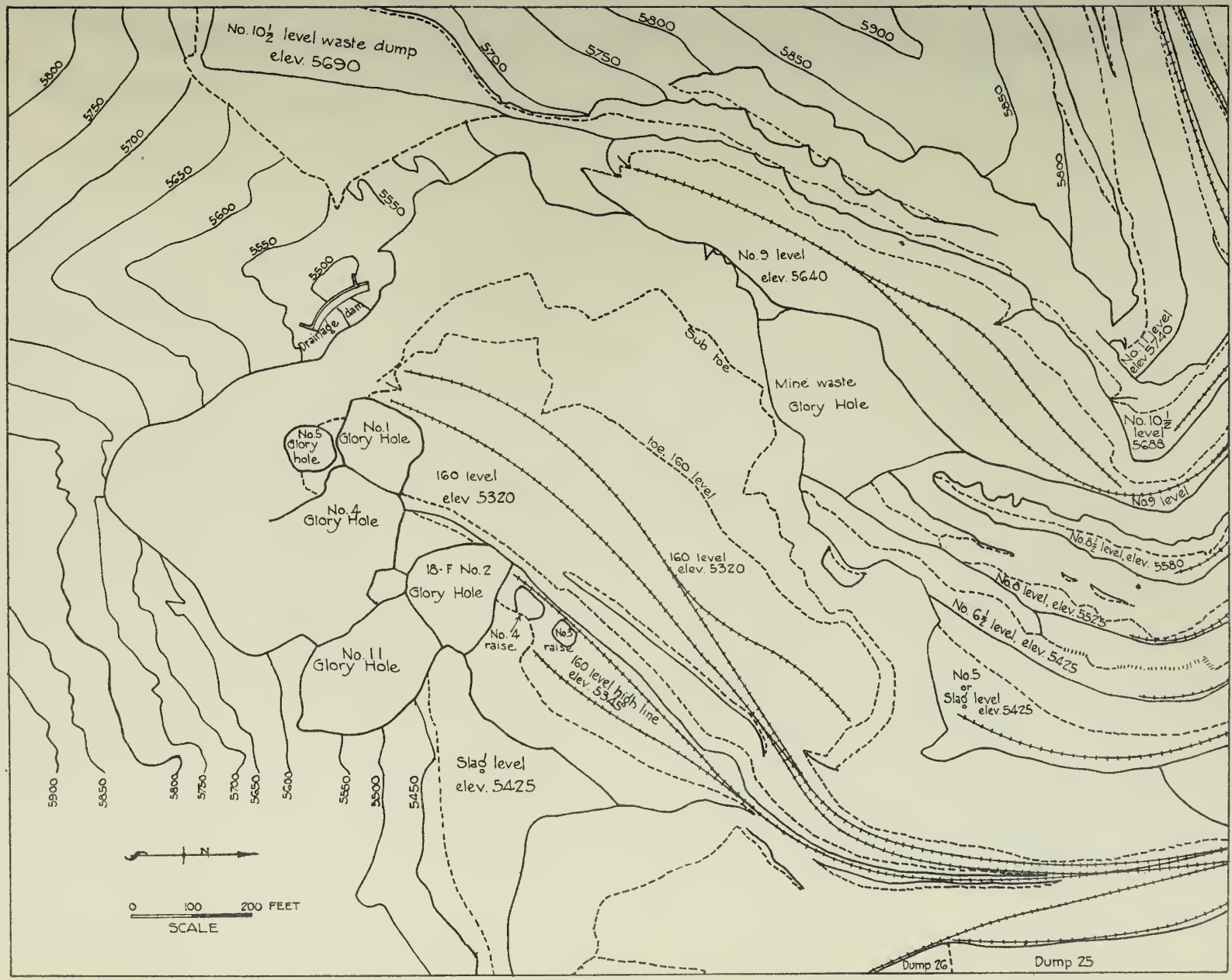


FIGURE 3.-SURFACE PLAN OF UNITED VERDE MAJOR STRIPPING OPERATIONS, JANUARY 1, 1925


MINE SURFACE PLANT - 500 FT. LEVEL
UPPER TOPAZ DUMP
LOWER TOPAZ DUMP
QUARTZ DUMP #27
300 LEVEL WASTE
OXIDE DUMP #24
"THE HAYNES"
CLARK FIELDS
#6 DUMP
#7 DUMP
#8 DUMP
#9 DUMP
#10 DUMP
#8 TAIL TRACK DUMP
#10 TAIL TRACK DUMP
PROPERTY LINE
HILL SPRINGS
WATER TANKS
V.T.&S.F.R.R.
PROPERTY LINE
#23 WASTE DUMP
OXIDE DUMP #25
300 FT. LEVEL
QUARTZ DUMP #26
WASTE GLORY HOLE
650' LEVEL
- ELEV 5325 -
3300 LEVEL
5270 LEVEL
#4 SHAFT DUMP
N
FIGURE 4.
PROGRESS MAP
UNITED VERDE COPPER COMPANY
JANUARY 1, 1926
SCALE: 1 IN. = 100 FT.
LEGEND:
Standard Gauge Track
Fresh Water Lines
Treated Water Lines
Air Lines
Oil Lines
Roads
Pipe Tunnels

MAJOR STRIPPING OPERATIONS

General

The major stripping operations were completed in October, 1927. The following review of the work done and the problems encountered is presented.

Ore above the 160 level consisted chiefly of oxides and quartz. Sulphide and schist ore were mined to a slight extent. The dense augite diorite was the main material stripped. On the surface the diorite was weathered and was easily broken down. Greenstone, porphyry, low-grade oxides and quartz, and approximately 1,600,000 cubic yards of slag made up the balance of the stripping material.

Development of Upper Pit

The upper pit had eight major levels (fig. 3). The bottom or 160 level had a bank 110 feet high, the next six above had banks approximately 50 feet high, and the bank on the top level varied in height from a few feet to 140 feet. Standard-gage railroad tracks connected each level by means of a system of switchbacks on the adjacent side of the hill (fig. 4). The switchbacks were planned with tailtracks on the pit end connecting with the levels. Waste dumps were maintained along the switchbacks and in nearby gulches.

Switchback Construction and Cost

In order to work on the various benches as planned, 12 standard-gage switchbacks were required between the 300 level, elevation 5,200, and upper bench, elevation 5,740 feet. Ninety-pound rails were laid on the four switchbacks between the 300 level and the 160 level, and 75-pound rails were laid on the upper switchbacks. Seventy-five pound rails were also employed on the benches for the loading track. Sixty-pound rails were used on the dumps. The maximum grade was 3 per cent. Material was hauled downgrade to the dumps. However, when it later became necessary to move material from the 160 level to a dump at a higher elevation, the grade on the fifth level was reduced to 2 per cent. The maximum curvature was 24°. All curves were tapered and compensated for grade. No. 7 switches were used exclusively. At one time approximately 7 miles of standard-gage track was being maintained.

General data on the switchbacks is tabulated as follows:

Total length of switchbacks	27,064 feet
Cost per mile excavating and grading	$42,700
Cost per cubic yard excavating and grading	$1.41
Cost per mile track material (75-lb. rail)	$18,000
Cost per mile laying track	$4,000

Equipment

Rolling stock used during the major stripping operation is tabulated as follows:

Rolling stock used in major stripping

No. used	Item	Type	Weight, size, or capacity
1	Steam shovel	Full revolving	8-cu. yd. dipper
2	Steam shovel	Standard	4-cu. yd. dipper
1	Steam shovel	Full revolving	¾-cu. yd. dipper
5	Steam locomotives	Switcher	82.5 tons
2	same	Switcher	53.5 tons
30	Cars	Air dump	25-cu. yds.
1	Spreader	- -	14-ft. spread
1	Locomotive crane	- -	100 tons
2	Track shifters	Peterson	
6	Gondolas	F. B. composite	50-ton
6	Flat cars	- -	50-ton
1	Tank car	- -	50-ton, 10,000-gals.

The full-revolving shovel was mounted on traction wheels running on parallel tracks laid with 130-pound rails. The 4-yard standard railroad shovels were first used on sections of standard-gage track and were supported rigidly by jack arms. These shovels were later equipped with caterpillar traction trucks, which provided greater flexibility and eliminated jacking delays.

The full-revolving shovel was placed on the lower bench of the 160 level. It was planned to make the lower bench 110 feet high, as that was the difference in elevation between the 160 level and the top of the slag dump. The 4-yard shovels were used on the upper benches. It was originally intended that work on the upper benches should be completed in sufficient time to permit the large shovel to remain on the 160 level and maintain a 110-foot bench. The small 3/4-yard shovel was used for excavation on switchbacks and roads.

The crew on each steam shovel except the small one consisted of an engineer, a craneman, a fireman, an oiler, and two to four pitmen.

The switcher-type locomotives were used for tramming. They were equipped with water leg-type tenders of 4,500 gallons capacity. The 82.5-ton locomotives were able to haul six loaded cars and the 53.5-ton locomotives could haul four cars. The locomotive crew consisted of an engineer, a fireman, and a switchman.

Both steam shovels and locomotives were originally equipped for coal burning, but later they were altered to burn oil, as it was found that a saving of 28.9 per cent of the cost per cubic yard trammed was made by using oil for fuel. The average cost of coal per ton for the period compared was $10.08 against $2.045 per barrel of fuel oil; 2.98 barrels of fuel oil was equivalent to 1 ton of coal. Superheated steam was also found to be advantageous for shovels and locomotives.

The 25-cubic yard air-dump cars were used for handling the loaded rock. Occasionally large pieces were loaded which would not pass through the raised door when dumping and which had to be blasted out.

The spreader was used for spreading the dumped material so that the track could be shifted.

The track shifters were particularly valuable for shifting track on the dumps as well as for loosening the track from the ballast when reclaiming.

The crane was equipped with clamshell and orange-peel buckets for handling coal and other material, as well as for excavating and grading. The crane was also used to assist in the heavy repair work on the shovels and cars and to pick up cars that were wrecked and turned over on the dumps.

Gondola cars were required for hauling coal and loose material. When the oil-burning equipment was installed the fuel oil was transported to the shovels in a tank car. Flat cars were used to handle construction material, repair parts for the shovels, drill steel, and blasting supplies.

Buildings and Shops

The buildings and shops consisted of the warehouse, office, locomotive house, machine shop, blacksmith shop, and car-repair shop. Storage tanks and a sand dryer were also required.

Water System and Treatment Plant

Sufficient water was obtained from several springs on the east slope of the Black Hills, and was distributed by gravity through pipes on the various benches. Treated water for distribution on the two upper levels was pumped to a storage tank above the top bench. A lime-soda-ash softener was first installed to treat the water, but did not eliminate foaming in the boilers. The foaming was due to a large amount of calcium sulphate present in the raw water which was changed to sodium sulphate in the treatment process. To eliminate this foaming an automatic lime-barium carbonate water softener with a capacity of 4,000 gallons hourly was installed. The hardness of the raw water, which varied from 14 to 30°, was reduced to 3°; chiefly by the elimination of the mineral constituents. The cost of treating the water by the lime-barium process averaged $0.066 per 1,000 gallons.

Drilling and Blasting

Drilling and blasting has always been the most serious problem at the United Verde pit. During the early stages of mining the ground was easily broken on the upper benches. The altered diorite was quickly prepared for blasting. Churn-drilling was mainly used during this period, with holes spaced 35 to 40 feet apart at a distance of 5 to 10 feet back of the edge. These holes were generally drilled 12⅓ per cent deeper than the height of the bank, and were loaded with 10 to 12 boxes of 35 per cent strength gelatin. Toe holes were used occasionally for removing hard ribs.

The favorable conditions did not persist. Below the altered zone the diorite was fresh, very hard, and dense, besides being blocky and fissured. The rate of churn-drilling dropped from 15 to 20 feet per shift to 3 to 5 feet per shift, and on account of the slips and seams occurring in the diorite, difficulty was experienced in keeping the holes straight. Churn-drilling was then abandoned and replaced by drilling with air machines. A combination of toe-holing with jack-hammers and drilling vertical holes with tripod-mounted drifters was finally adopted (fig. 5) after toe-holing alone proved impractical. Shooting with toe holes alone caused the bank to be undercut and did not secure fragmentation, which resulted in delaying the shovels while bowlders were being blasted. A general comparison of costs during these three periods is as follows:

Comparison of costs of drilling and blasting methods

	Churn-drilling, per cu. yd.	Toe-holing, per cu. yd.	Combination method, per cu. yd.
Labor	$ .0467	$ .0631	$ .1132
Explosives	.0259	.0438	.0683
Total	.0726	.1069	.1815
Average cubic yards per 8-hour shift loaded by shovels..	800	530	758

On the lower bench or 160 level the slag and material from the oxidized zone were removed very rapidly, whereas progress on the upper levels was slow because of the difficulty of breaking the unaltered diorite; this disarranged the schedule of operations. As the large shovel crowded the work on the upper benches, a consolidation of the benches on the west end resulted, so that on the west side of the pit the height of the bank above the 160 level reached 315 feet. In order not to shut down the large shovel on the 160 level or move it to an upper level the ground beneath this high bank had to be broken. Coyote blasting was then used to break the ground. By this time most of the material from the oxidized zone had been removed, and the material in the bank was all barren diorite. Large blasts were therefore practical, as not much of the material would become mixed. Six coyote blasts were set off to break material on the 160 level.

Two powder foremen, each assisted by a jigger boss, supervised the drilling and blasting; one foreman worked on the lower bench and the other on the upper benches. The drilling and blasting crew on the lower bench consisted of about 25 men. On the upper benches 50 to 60 men were needed.

The principal methods of breaking ground at the United Verde upper pit--churn-drilling, combination toe-hole and vertical drilling, and coyote blasting--will be considered in detail.

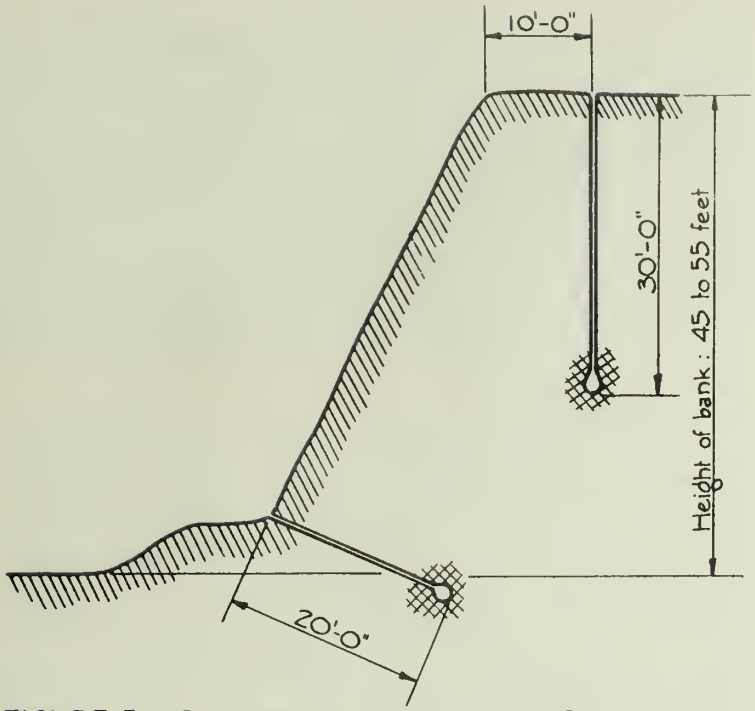


FIGURE 5.- SECTION OF BANK ON UPPER LEVELS SHOWING VERTICAL AND TOE HOLES

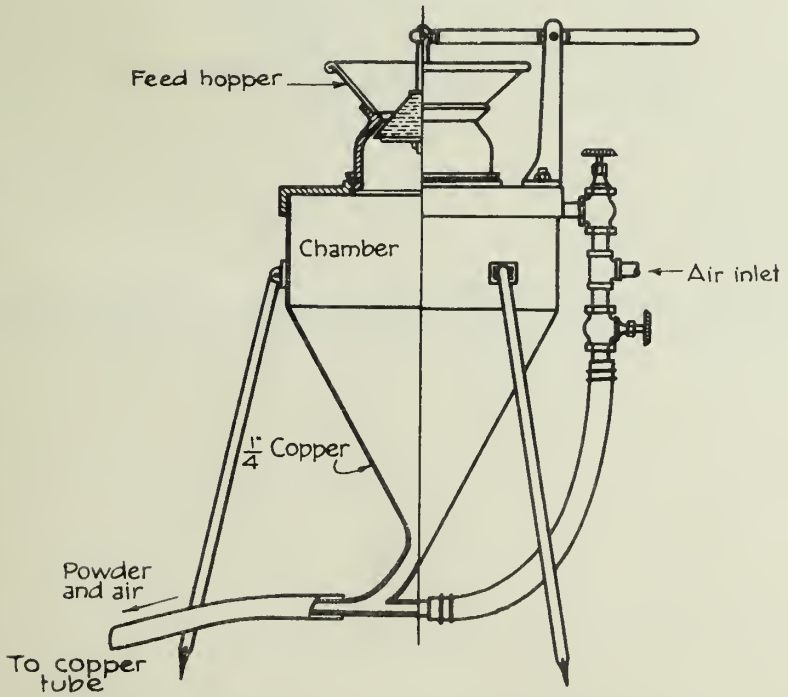


FIGURE 6.- UNITED VERDE POWDER LOADING MACHINE

Churn Drilling and Blasting on the 160 Level. - For breaking the softer ground on the lower level three steam churn drills were mainly used. Six-inch holes were drilled 10 to 15 feet deeper than the height of the bank. Holes were cased with 6-inch pipe in loose ground. The drilling speed varied from 5 to 30 feet per shift, depending on the character of the material. The holes were placed close to the edge of the bank and spaced from 30 to 50 feet apart. The distances from the bottom of the hole to the toe of the cut varied from 55 to 95 feet. These distances and the character of the ground governed the size of the powder charge, which ordinarily ranged from 4,000 to 8,000 pounds. In a few instances the burden on the churn-drill holes was reduced with toe holes, but on account of inability to trim the bank thoroughly, this was considered unsafe.

Hot ground was encountered in certain areas on the 160 level which were churn-drilled. The temperature in some of these holes varied from 120 to 240° F. It was necessary to cool these holes before loading. In some cases if the hole remained for two or three days before loading it would cool off sufficiently to permit loading; in others it was necessary to run a stream of water into the hole for as much as 24 hours or more. In all holes 35 per cent strength gelatin dynamite was used, as it was considered less sensitive to deterioration from the heat of the rock. A temperature of 120°F. was considered safe for loading. To permit the men to keep away from the collar of the hole when loading, a wooden trough was designed by which powder and sand tamping could be pushed into the hole from a distance. Generally four No. 8 electric caps were placed in the large holes and fired from a 440-volt line. Seldom more than three holes were shot at a time. The cost of churn-drilling for 1936 is as follows:

	Per foot
Labor	$1.97
Supplies	.65
Fuel	.68
Repairs	.98
Miscellaneous ...	.26
Total	$4.54

Average footage drilled was 9.2 feet per shift.

A drill crew consisted of three men - a driller, a fireman, and a helper.

Combination Machine Drilling and Blasting. - The general data referring to breaking ground by machine drilling and blasting is tabulated as follows:

Breaking ground by machine drilling and blasting

Item	Toe holes	Vertical holes
Kind of drill used	Jackhammers	Large Leyners
Depth of hole	18-20 ft.	21-30 ft.
Feet drilled per hour	3.5	3.7
Drilling crew	1 man	2 men
Labor cost per foot	$0.22	$0.42
Percentage of holes lost ..	20	35
Spacing of holes in line ..	6-15 ft.	12.5 ft.
Powder charge	100-250 lbs.	75-150 lbs.
Kind of powder	50 per cent gelatin and Quarry Special No. 1	Quarry Special No. 1 and 50 per cent gelatin
Steel used	7/8-in. quarter-octagon	1¼-in. hollow-round
Gage of steel, starter	2-1/4 in.	2-3/4 in.
Gage of steel, finisher ...	19/16 in.	21/16 in.
Length of starter	2-1/2 ft.	2-1/2 ft.
Length of finisher	20 ft.	30 ft.
Number of changes	12	19

Jackhammers, which are light and easy to handle, were used for drilling toe holes. In drilling vertical holes the Leyners were suspended by a block and tackle from a tripod made from 2-inch pipe. On the heavy machines two men were required to handle the larger steel. When two parallel rows of vertical holes were shot at the same time the holes were staggered.

An attempt was made to drill toe holes with Leyners mounted on tripods and also with heavier jackhammers. However, on account of the time required to set up, the greater loss when one of the larger holes had to be abandoned in the fissured ground, and the extra labor required, the use of the heavier machines was abandoned in favor of the light jackhammers.

Compressed air was obtained from the main mine compressors on the 500 level and was delivered to the pit at 85 pounds pressure through a 4-inch pipe line about one-half mile long.

Both vertical and toe holes were heavily chambered to receive the required amount of explosives. The quantity of explosives used in chambering the holes varied from 10 to 15 per cent of the charge. During the first stage of operations the chambered holes were loaded with stick explosive, and the cartridges were pushed into the chamber with a wooden rod. This method was slow, particularly in fractured ground; also the miners were inclined to use unsafe methods in forcing the cartridges through the ragged portions of the holes that resulted from springing.

Experiments were made in loading granulated powder with compressed air, and the powder loader shown in Figure 6 was designed and adopted. Whereas it required 20 minutes to load 50 pounds of stick explosive, the same amount of granulated powder could be loaded in 60 seconds. The powder loader consisted of

a sealed copper chamber fitted with a hopper through which the powder was placed in the chamber. Compressed air was then admitted into the chamber; the powder was forced down into a flexible rubber tube, from which it was carried by a jet of compressed air through a copper loading tube. This loading tube was placed in the hole to be loaded, the end of the tube resting a few inches from the bottom of the chamber. As the chamber filled up, the tube was withdrawn. The experienced operator was enabled to determine when the chamber had been filled by the change in rate of withdrawing the tube. The chamber of the loader was designed to hold 50 pounds of granulated powder. Quarry Special No. 1 powder was found best suited for use in this machine. Before being placed in the container, the powder was put through a 4-mesh copper screen, so that no lumps large enough to clog the loader would be admitted.

Air loading of granulated powder is generally considered unsafe by the powder manufacturers. However, during the three years the method was used at the United Verde pit not even a slight mishap occurred, whereas several serious accidents resulted from the former method of loading.

No. 8 electric detonators were used exclusively for detonating the charges. Two caps were placed in each hole, one previous to the charge and the other after the chamber was about three-fourths filled. Current at 440 volts from the main power line was used for igniting the detonators. Between 20 and 30 holes were fired at one time.

Although the combination-method toe and vertical holes gave better fragmentation, an enormous quantity of bowlders was still present in the blasted rock. If it was possible for the shovel to set the bowlders aside, they were later block-holed. In many instances the bowlders had to be "dobied" before the shovel could operate. Approximately 20 per cent of the total powder used was for secondary blasting.

Coyote Blasting. - Coyote blasting was confined to the 160 level. Six of these large blasts were fired during the major stripping operations. The high banks that were not accessible for churn drilling could be broken by means of toe holes with a fair degree of success in soft ground. In hard diorite, however, the banks could not be broken in this manner. A round of toe holes drilled with considerable danger beneath the high bank on the west side of the pit merely undercut the bank and furnished less than half a shift's digging for the large shovel. The fresh diorite in high banks was also too hard to be broken by means of churn-drill holes. In order to break the high bank with safety and in sufficient quantities, coyote blasting was deemed to be the only practical method.

The development work of the coyote blasts consisted of driving the necessary adits and sinking the powder pockets at the desired distances from the bank. For the first three blasts, work was started from the face of the bank. However, this required that the bank be cleaned up before development for another blast could be started. For the last three blasts an old adit behind the bank was utilized, and as soon as one blast had been fired development for a new blast started. The drifts were 4½ by 6 feet in cross section. Powder pockets were

5 by 5 feet in section, and were sunk to a depth of 20 to 30 feet below the pit grade at intervals of 30 feet in the drifts. The distance from the pockets to the face of the cut varied from 78 to 135 feet in the different blasts. Upon the completion of the development work, the pockets were wired up and the lead wires protected by being encased in old rubber hose. Four No. 8 electric detonators in a case of gelatin dynamite were placed in the bottom of each pocket. The powder was unloaded from box-cars at the mouth of the tunnel and transported to the pockets on specially constructed wooden cars. After the charge was loaded into the pockets, the powder was covered with paper and sand tamping. Back-filling of the tunnel was started as soon as the pockets had been loaded.

After the first coyote blast, heat radiating from the mine fire and resulting high humidity were encountered in one of the end pockets. The rock temperature in this pocket was 152°F. In the other pockets the heat varied from 112 to 124°F. During development work it was necessary to cool the drift by means of blowers and piping. Tests on explosives and detonators were made under the conditions existing in the end pockets. It was found that black powder absorbed moisture and became useless; 35 per cent strength gelatin dynamite lost some sensitiveness, and 50 per cent gelatin dynamite appeared unaffected. The tar covering on the electric caps was melted, but the cap tested satisfactorily, as did Cordeau Bickford. To overcome the moisture, the pockets and drift adjacent to the hottest ground were gunnited. This did not reduce the temperature. Before loading the powder charges the pockets were lined heavily with asbestos to keep the powder from direct contact with the hot ground.

Black powder was used in the first three blasts. The first blast was in the softer material along the contact, and was considerably overshot. The second and third blasts were undershot, as the black powder did not have sufficient shattering force to break the hard ground effectively. To obtain better shattering of the rock, Quarry Special powders with a higher disruptive effect were used. The speed was further increased by the addition of higher explosives; 60 per cent strength ammonia dynamite was used in the last blast. The ground was satisfactorily broken in the last three blasts with a considerable decrease of secondary blasting. Figure 7 shows the plan of blast 6 and Figure 8 shows the section through No. 9 pocket for blast 6.

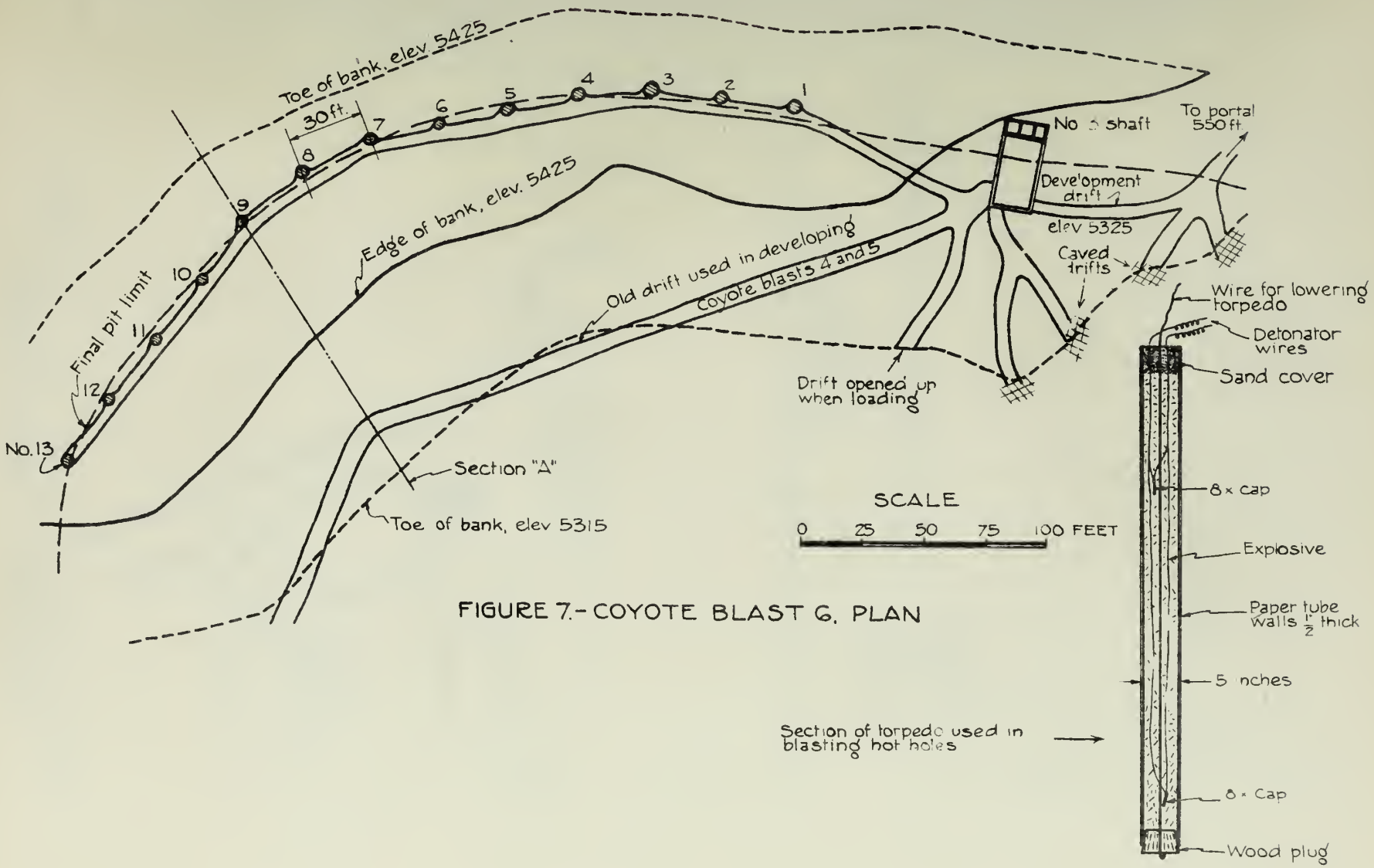


FIGURE 7.- COYOTE BLAST 6, PLAN

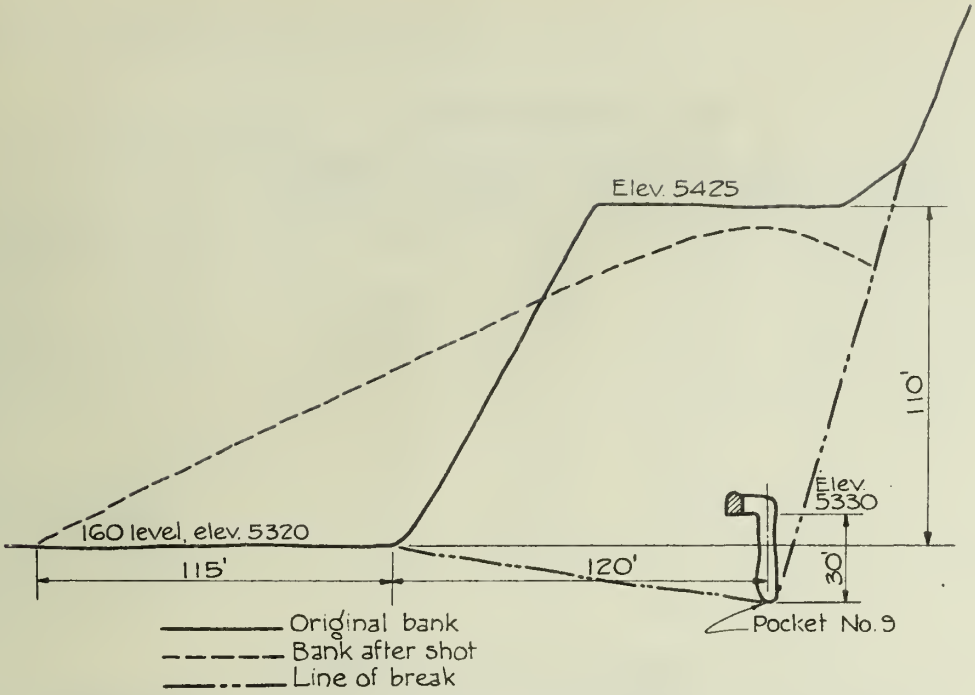


FIGURE 8.–SECTION "A" THROUGH POCKET NO. 9, COYOTE BLAST 6

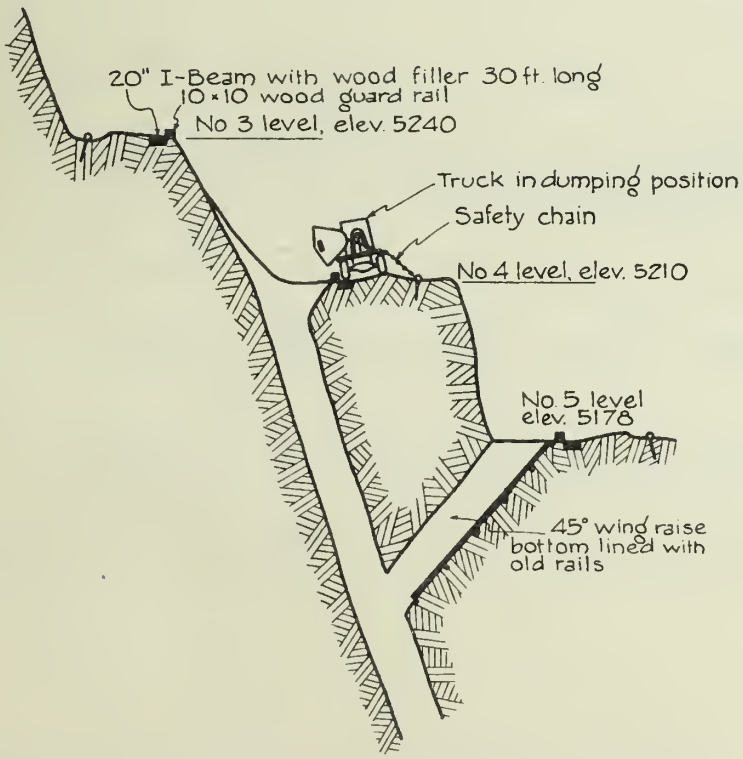


FIGURE 10.– TYPICAL SECTION THROUGH NO. 3 RAISE SHOWING DUMPS ON THREE LEVELS

Table 1 gives detailed information on each of the six coyote blasts.

Table 1. - Coyote blasting data

	Blast No. 1	Blast No. 2	Blast No. 3	Blast No. 4	Blast No. 5	Blast No. 6
Number of pockets	7	6	14	10	5	13
Average distance between pockets, feet	30	30	30	30	30	30
Total development drifting and sinking, feet	410	485	910	686	225	800
Average distance of pocket to toe of bank, feet	78	100	133	65	61	100
Average vertical height above pocket, feet	121	167	189	146	120	120
Average height of bank, feet	(1)	(1)	(1)	(1)	100	100
Average burden on each pocket, cu. yds.	5,000	7,800	9,800	6,000	5,230	9,240
Black powder used, pounds	100,000	95,400	250,000	- -	- -	- -
Quarry Special No. 6 used, pounds	- -	- -	- -	91,700	- -	200,000
Quarry Special No. 4 used, pounds	- -	- -	- -	- -	45,000	- -
60 per cent ammonia dynamite used, lbs.	- -	- -	- -	- -	5,000	30,000
50 per cent gelatin dynamite used, lbs.	13,050	3,500	9,900	11,650	- -	3,250
35 per cent gelatin dynamite used, lbs.	- -	- -	- -	- -	4,000	- -
Total pounds of powder used	113,050	98,900	259,900	103,150	54,000	233,250
Cubic yards broken	140,000	135,000	259,000	105,000	54,000	200,000
Cubic yards per pound of explosives	1.24	1.36	1.00	1.02	1.00	0.86
Cubic yards per foot developed	394	278	285	153	240	250
Costs per cubic yard:						
Development, labor and supplies	$0.0850	$0.0827	$0.0502	$0.1297	$0.1400	$0.0407
Explosives	0.0783	0.0697	0.0915	0.1065	0.1360	0.1272
Total cost per cubic yard	0.1633	0.1524	0.1418	0.2362	0.2760	0.1679

1 Pockets under slope only.

Shovel Operation

On the 160 level the first cut with the large shovel was parallel to the face of the slag dump, and the cars were loaded on the opposite side from the bank and on a higher grade than the cut. After a considerable amount of slag had been removed, the shovel was headed directly into the ore body and a thorough cut made through the entire extent. The loading track was carried on one side of the cut about 15 feet above the level of the shovel, and ground was filled in on one side so that the track could be extended as the cut advanced. After the first cut was completed the shovel began widening out on the north and west sides. As the cut became wider it was necessary to shift the loading track to the same level as the shovel.

On the upper benches where the slope of the ground was not too steep a narrow cut was made on the grade of the bench and track was laid for loading. Ground to be loaded by the 4-yard shovels could be broken down without covering the track; the level was established as soon as the first cut was completed.

When the slope was too steep, it was necessary for the 4-yard shovels to cast the broken material over the bank to make room for the track and to establish the level. This material was picked up on a lower level, generally the 160. As soon as a level was established, the shovel was moved back to one end. Ground was broken ahead of the shovel, which advanced the entire length of the bench. After each cut, the loading track was shifted to parallel the face of the cut. With the 4-yard shovels the top level was completed first and lower benches were completed successively.

Two bench foremen had direct charge of the work. One foreman looked after the shovel and locomotive operations in the pit and on the dumps of the 160 bench. The other foreman had charge of the two shovels on the upper benches and of the tramming operations. On account of the short haul to the switchback dumps on the upper benches, a switch tender was not required. However, on the 160 bench a switch tender was required at the entrance to the pit and also on the dump approach, in order to control the movement of trains.

During the time all three shovels were in operation two shifts were maintained - a day shift from 7.30 a.m. to 5 p.m. and a night shift from 6 p.m. to 2 a.m. After the 4-yard shovels were removed from the upper benches, the 300 shovel was worked on three 8-hour shifts.

Disposition of Material. - On the upper benches the material consisted chiefly of diorite, with a small amount of low-grade quartz ore on the two benches above the 160 level. Most of the ore was obtained on the 160 level. The material encountered was classified and disposed of as follows:

Classification and disposition of material

Material	Grade	Class	Destination
Quartz	Over $1.25 in Ag and Au	Converter ore	To smelter or stock pile for smelter.
Quartz	Below $1.25 in Ag and Au	Possible converter ore	To stock pile.
Oxide	Over $6	Direct smelting ore	To smelter or stock pile.
Oxide	Over $1.25, below $6		To stock pile.
Oxide	Below $1.25	Leach ore	To leach dump.
Sulphide	Over 3% copper	Direct smelting ore	To smelter.
Sulphide	Over 1%, below 3% Cu	Possible smelting ore	To stock pile.
Sulphide	Below 1% copper	Leach ore	To leach dump.
Schist	Over 3% copper	Direct smelting ore	To smelter.
Schist	Over 1%, below 3% Cu	Possible milling ore	To stock pile.
Schist	Below 1% copper	Leach ore	To leach dump.
Diorite, slag, greenstone, porphyry	Barren	Waste	To waste dump or underground mine for filling.

The first material shipped to the smelter was transported directly from the pit to the smelter over the Verde Tunnel & Smelter Railroad. The Hopewell haulage tunnel constructed on the 1,000-foot level ran through the footwall of the ore body. By dropping the ore from the pit through raises and hauling it out of the tunnel it was estimated that a saving of 31 cents a ton could be made in transporting material from the pit to Hopewell. Therefore two raises were completed in 1922, one for direct-smelting ore, the other for converter ore. These raises were brought up in the footwall adjacent to the pit. A track was extended along the sides of the raises so that the loaded cars could be dumped directly into them. On the 900 level the raises were equipped with grizzlies and blasting chambers for reducing the larger material. Between the 900 and 1,000 foot levels, storage bins of 1,000-ton capacity were cut out and provided with pneumatic doors and chutes. Forty-ton standard-gage cars are used in the Hopewell tunnel.

During the major stripping operations the United Verde mill had not been completed. As most of the schist encountered was of high grade it was shipped as direct-smelting ore.

Stock piles were established at convenient locations adjacent to the pit and at the mouth of Hopewell tunnel. Material which could not be smelted at the time and low-grade material which might have future value were placed on these stock piles. At Jerome dump 25 was made for the higher-grade oxides and dump 24

for the lower-grade oxides. High-grade quartz was placed on Hopewell dump 34. Lower-grade quartz was placed on Jerome dump 26 and Hopewell dump 33. The low-grade sulphide stock pile was at Hopewell.

A waste glory hole was maintained on the surface in the diorite and connected with a raise for the distribution of waste to be used as filling in the underground mine. This waste-transfer raise intersected with the pit levels. The underground mine was assured of a sufficient quantity of waste for filling the stopes as long as the open-pit operations continued. However, the waste delivered to the underground mine represented a very small part of the total waste handled, being about 150,000 cubic yards per year. Part of the waste was placed on the switchback dumps, and the switchback tracks were moved out from the hill as the dumps were built up. Most of the waste was placed on four dumps filling up adjacent gullies. The height of the dumps varied from 50 to 200 feet. The average haul was about 4,000 feet and the maximum 7,000 feet.

The proper destination of the material was determined by bank samples obtained before the ground was blasted and by samples taken of the broken material. If classification of the material was doubtful, the cars that had been loaded were set out until the assay was returned. All cars were grab-sampled after loading so that a record could be obtained of the material, especially of that which was placed on the stock piles. All pit samples were sent to the main assay office for assaying.

Costs of Major Stripping Operation

The cost of stripping was deferred to be absorbed later as the ore was made available. That is, the ore later recovered would carry a charge for removing the waste above it. To the direct charge for mining, an estimated charge of $1.50 was added to pay for the stripping removed.

Direct operating costs for 1925 are given in Table 2.

It may be noted in the summary of direct operating costs that the charge for repairs is very high. The United Verde pit probably has harder rock to handle than any other pit. Work in this hard rock resulted in severe strains to the digging equipment, which accounts for the high repair cost.

Table 2. - Summary of costs

United Verde major stripping operations — Year 1925

	Cu. yds.
Total stripping removed in 1925	1,647,469
Total ore removed in 1925	165,084
Total material removed in 1925	1,818,093
Dry tons of ore mined in 1925	272,005 tons

Direct operating costs per dry ton ore mined and per cubic yard of stripping

	Labor	Comp. air drill and steel[1]	Power cost	Fuel cost	Explo-sives	Re-pairs[2]	Sup-plies	Misc.	Total
Stripping (per cu. yd.)									
Drilling and blasting .	$.1012	$.0361		$.0029	$.0871		$.0100	$.0037	$.2410
Loading	.0541		$.0005	.0430		$.1083	.0061	.0022	.2142
Transportation	.0550			.0447		.1011	.0024	.0081	.2113
Dump maintenance ..	.0381	.0002		.0006		.0016	.0057	.0122	.0584
Total stripping.	.2484	.0363	.0005	.0912	.0871	.2110	.0242	.0262	.7349
Mining (per ton)									
Drilling and blasting .	.0693	.0234		.0008	.0459		.0073	.0024	.1491
Loading	.0403		.0030	.0165		.0528	.0036	.0013	.1175
Transportation	.0349			.0177		.0437	.0058	.0015	.1036
Total mining ...	$.1445	$.0234	$.0030	$.0350	$.0459	$.0965	$.0167	$.0052	$.3702

1 Repairs to drills included.
2 Includes labor and repair supplies.

General operating data for 1925 are tabulated as follows:

Shovel operation

	4-cubic yard shovels	8-cubic yard shovels
Total 8-hour shifts	1,244.6	747.5
Cubic yards per shift	784	1,001
Cubic yards per barrel of fuel oil	64.7	50.8
Barrels of fuel oil per shift	12.1	19.7
Per cent time of loading	46.4	49.1
Per cent time of operating delays	38.6	36.9
Per cent time of general delays	15.0	14.0

Locomotive operation

Total 8-hour shifts	4,485.3
Cubic yards per shift	357.6
Cubic yards per barrel of fuel oil	46.7
Barrels of fuel oil per shift	7.66

Blasting

Pounds explosives per cubic yard broken	0.648

Miscellaneous

Total man-shifts	154,191
Cubic yards (ore and stripping) per man-shift	11.79
Total labor cost per cubic yard	$.4081
Repair cost (labor, supplies, power, etc.) per cu. yd.	$.2325

LOWER PIT OPERATIONS

Operations below the 160 level are confined chiefly within the main ore body. The entire area of the ore body is exposed on the 300-foot mine level. Below the 300 level, ore will be left in the pit wall to be removed later, and between the 300 level and the 160 level considerable stripping will have to be removed in order to maintain a safe working slope. In the harder material an attempt is being made to maintain a 60° slope, while in the softer material a 45° slope is necessary. Scarcely any oxide remained below the 160 level. The massive sulphides form the chief ore material, and on the sides of the sulphides, mineral-bearing schist for milling and quartz for converter ore are obtained. Low-grade greenstone, porphyry, schist, jasper, and barren diorite are more or less on the margin of the ore body. Throughout the mass of the ore body underground workings are encountered. The old square-set stopes are generally filled. However, in some areas the old stopes were abandoned when the fire broke out, and

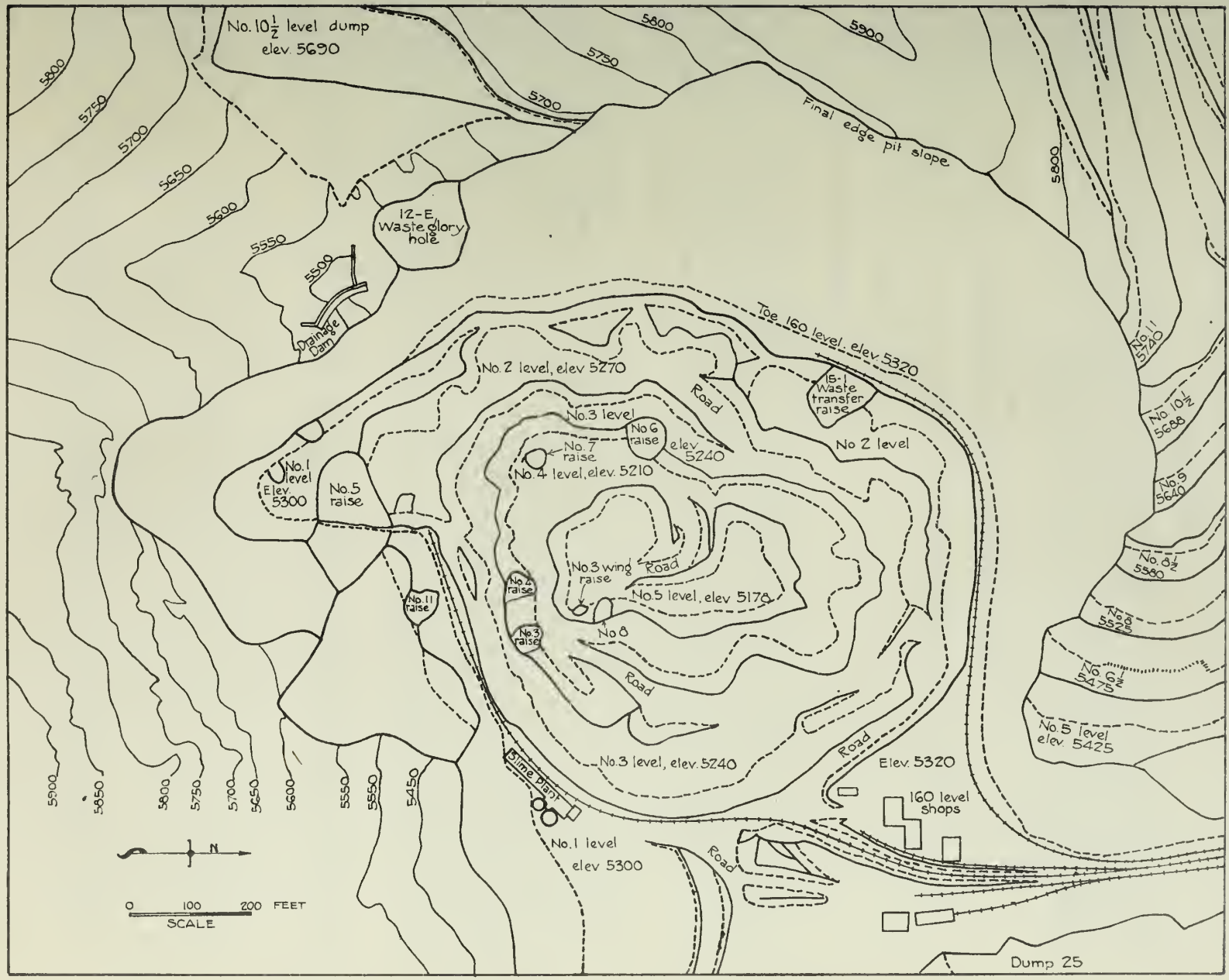


FIGURE 9.- PLAN OF UNITED VERDE PIT, JULY 1, 1929

the upper floors could not be filled. In operating in the lower pit, these old workings have to be considered in order to avoid mishaps.

Development

As has been noted before, the original plan for removing material below the 160 level was by glory-holing. In 1924, after a portion of the ore body on the 160 level bench had been exposed and made accessible, several glory holes were started. However, they did not prove practical for the following reasons:

(a) Inability to drive raises in the desired locations on account of the underground fire and existing stopes.

(b) Dilution of ore and inability to sort material properly.

(c) Indefinite production.

(d) Operating hazards.

The United Verde mine fire extended more or less through the entire ore body above the 800 level, except in the extreme south end. Although certain parts of the ore body were accessible on various levels, the driving of a continuous raise from the 1,000-foot level was practically impossible except in the footwall. In the small glory holes that were started in the sulphide area, transfer of material on intermediate levels was necessary to avoid the fire areas.

In the several attempts to drive raises throughout the main ore body it was necessary to open the fire areas which had been sealed off. In these areas water had to be sprayed on the timbers in a few instances to prevent combustion. Double crews had to be maintained to keep the work going, and it was necessary to have cool air blowing on the men continuously. In several places the heat of the ground necessitated wrapping the gelatin dynamite in asbestos when loading holes. Moreover in certain areas the burnt ground would not stand up, as the sulphides had been reduced to a powder by the action of the fire. After a few attempts were made and a serious accident occurred by the igniting of the dust when blasting,[3] further work was abandoned.

In driving raises through the old stopes it was impossible to hold the dried-out filling satisfactorily. Raises therefore had to be driven through the solid ground.

The ore body of the United Verde mine is very complex. High-grade sulphides, schist, and quartz are adjacent to each other or to low-grade materials. High and low grade areas also occur in the mass of sulphides. Glory-holing was started around No. 3 and No. 4 raises (fig. 9). Alternate seams of high and low grade porphyry and schist surrounded the collars of these raises. In blasting the

3 Gardner, E. D., and Stein, Edmund, Explosibility of Sulphide Dusts in Metal Mines: Repts. Investigations, Serial 2863, Bureau of Mines, 1928, 11 pp.

material the different grades became mixed, as it was impossible to blast advantageously in the confined limits of the glory holes. If it was possible to blast different materials separately, the destination of the material as it was removed on the 1,000-foot level was changed to correspond with the different classes. However, part of one class of material stuck to the sides of the raise and in the bin, causing a dilution of the material as the raise was changed to high or low grade material. In addition, the separation of stope fill presented difficulties, and the old timbers encountered had to be pulled out of the glory hole in order not to clog the raises and chutes.

No definite production could be depended upon in a system of glory holes, as the material available would be more or less controlled by the working condition of the various glory holes. It would be impossible to work a certain glory hole faster in order to obtain the desired material in that area without interfering with work in adjacent glory holes. Greater difficulties in keeping the connecting raises clean and open would also be expected, as large rocks would be thrown into the raises when blasting.

Glory-holing is more or less hazardous because men have to work on slopes around loose rock. In the United Verde pit men would also be subjected to the added hazard of working under high walls on the sides of the pit and of not being able to leave the working place conveniently if loose material sloughed off of the high bank. Also above the fire area men would have to work on hot ground through which gas and smoke seeped from the mine fire below.

These conditions prevented the mining of the lower pit by glory-holing, but in the early part of 1925 a plan was devised by Mr. W. V. DeCamp, now assistant general manager of the United Verde Copper Co., by which the lower pit would be mined by small shovels working on low benches. The materials were loaded into trucks which dumped into any one of several transfer raises extending to the 1,000-foot mine level. The disadvantages of the glory-hole method were entirely overcome by this change.

Several additional raises were constructed in the porphyry footwall away from the fire area. An old mine shaft was cleaned out and utilized. Except for cutting around a transfer raise it is possible to use one raise constantly for one class of material, and dilution in the raises is practically eliminated. By the use of trucks operating on the levels, material can be transported conveniently from any part of a level to any raise. Figure 10 shows the arrangements for dumping into a transfer raise from three different levels. A plan to use cars operating on narrow-gage track was given up as it did not provide maximum flexibility. Steep grades connecting the various levels are necessary, and it is possible for motor trucks to haul from one level to another.

The small shovels are able to sort material very well. The shovels can also be moved around so that drilling and blasting can be done advantageously. The big problem in the lower pit operations is the sorting of the ore. This is first accomplished by proper drilling and blasting, and later by sorting and thorough cleaning up of loose material when digging. Softer material is generally

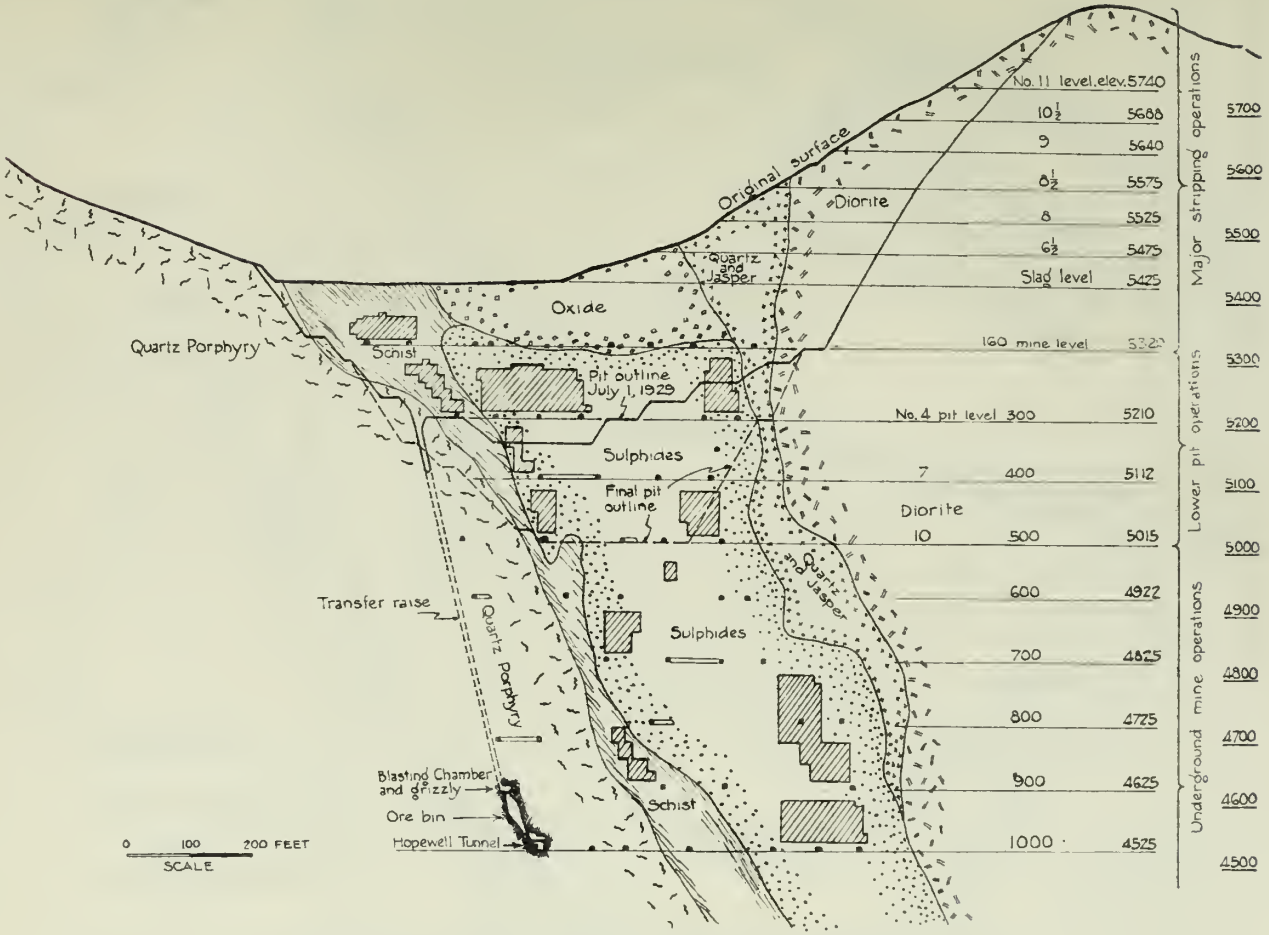


FIGURE 11.- TYPICAL SECTION OF UNITED VERDE PIT

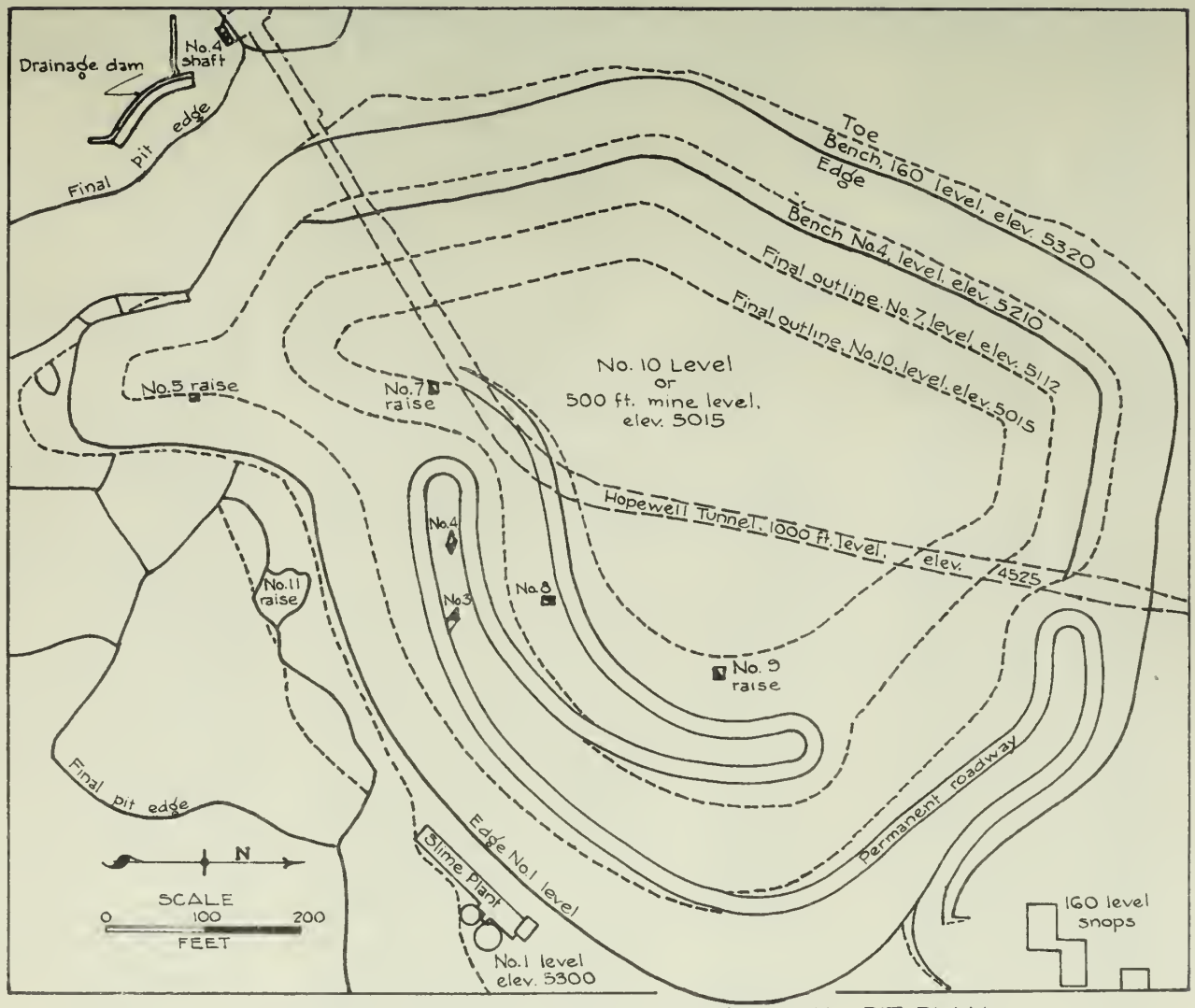


FIGURE 12.-UNITED VERDE LOWER PIT, FINAL PIT PLAN

removed first, leaving the harder material clean. Old timbers are set aside when encountered and are later loaded into trucks and hauled out of the pit. By planning the work in advance the levels can be developed properly to assure a constant production of the desired material. The small shovels can be quickly moved from one area to another or used between levels if necessary.

Maximum flexibility is possible in this method of operation. Shovels can be worked on any exposed bank. The trucks can be routed from any position to any raise. By this flexibility the pit is enabled to supply any material called for.

By operating on low banks much danger is removed. Men on the shovels and trucks are off of the hot ground and in motion so that they do not have to remain in the gassy areas. Most drilling in fire areas or gassy places is done with churn drills, which likewise can be moved anywhere on the various levels. With the churn drills it is not necessary for the drillers to remain steadily in the bad places.

The lower pit between the 160 and 500 levels is divided into twelve benches varying from 25 to 33 feet in height (fig. 11). At the present writing the first level below the 160 level, or No. 1 level, had been completed. Operations are now being conducted on Nos. 2, 3, 4, and 5 levels, the last of which is 33 feet below the 300 level at an elevation of 5,178 feet.

No. 1 level was first developed by a through cut made by one of the standard 4-cubic-yard steam shovels. The material, which was loaded into dump cars and hauled out, had to be disposed of as encountered; no flexibility was provided. This through cut was started several hundred feet outside of the pit limit on the 160 level on a 3 per cent downgrade and was cut alongside Nos. 3 and 4 raises. The first small shovel was then put into operation. Four light aviation-type army trucks were first used, until the practicability of the plan could be established. These light trucks were unable to pull upgrade. The cut, in going down to No. 2 level, was made in several 7-foot layers, and the shovel loaded the truck on the bench above. This made an upgrade necessary. Before No. 3 level was developed, heavy six-wheel, four-wheel-rear-drive trucks were obtained which could easily pull a load up a 10 per cent grade.

In general, when starting a new cut an attempt is made to confine it within one class of material. The cut is started toward one of the transfer raises so that when it reaches the elevation of the lower bench it will be adjacent to the transfer raise and the shovel can cast directly into the raise. Material can then be disposed of without hauling uphill. As soon as the level is established, work is concentrated in the direction of another transfer raise, and the material is put in the first raise. If a different class of material is encountered, the raise is cleaned out to accommodate that material. Levels are developed in ore first. The first cut is temporary, but a permanent roadway is to be made on the footwall side extending from the 160 level to the successive levels. This road has been completed to No. 4 level (fig. 12). The permanent road is to be on a 10 to 12 per cent grade and will wind back and forth on the footwall side of the pit. As the permanent road can not be conveniently situated with respect to the transfer raises, it is impossible to develop new levels from the roadway.

Equipment

The equipment used at the United Verde lower pit is tabulated as follows:

Equipment employed at the lower pit

Item and number used	Type	Description
4 Electric shovels	Full revolving	1¾ cu. yd. dipper.
10 Trucks	6-wheel, 4-wheel rear drive	4-cylinder, 10-ton capacity.
1 Truck	do.	6-cylinder, 10-ton capacity.
6 Tractor trucks	2 front wheels, caterpillars on rear	6-cylinder, 10-ton capacity.
2 Trucks	4-wheel	4-cylinder, 5 to 6 ton capacity.
1 Truck	4-wheel	6-cylinder, 3½-ton capacity.
1 Truck	4-wheel	6-cylinder, 2-ton capacity.
1 Truck	4-wheel	6-cylinder, 1-ton capacity.
1 Crane	Full revolving traction	Gasoline-powered.
1 Road scraper		Tractor-powered.

Electric-powered shovels, for which electricity is obtained from the smelter power plant, are used in the lower pit. A considerable saving is made by the use of electricity, as this form of power is more adaptable to the working conditions. These shovels are operated on 3-phase, 440-volt, 60-cycle alternating current. The shovel is equipped with a 100-horsepower hoist motor, 40-horsepower boom motor, and 60-horsepower swing motor. The shovels were originally equipped with 40-horsepower swing motors, but they did not stand up under severe operation. The shovels are mounted on caterpillars having a 36-inch tread and an overall width of 13 feet, 6 inches. The efficiency of the shovels was greatly increased by equipping them with manganese steel caterpillar pads. Other improvements have been made at the United Verde shops such as changing the propelling clutches and installing an automatic air valve for operating the clutch brake ram. The shovel crew consists of three men - an engineer, an oiler, and a pitman. During 1928 an average of 450 cubic yards, or approximately 1,050 tons, per shovel shift was obtained. Power consumption amounted to 0.974 kilowatt hour per cubic yard loaded.

The tramming trucks and tractor trucks are equipped with a standard gravity side-dump body, designed and built by the United Verde Copper Co. By releasing the latches the load can be dumped very quickly when the truck is slightly tilted. The body is built of heavy metal plate, channels, and angle bars riveted together. A double-plate bottom with a wood filler is provided for reducing the shock caused by dropping the heavy material into the truck. The body weighs 3 tons and has a capacity of 5 cubic yards water level, or 3½ cubic yards solid material. The load varies between 7 and 10.5 tons, depending on the character of the material handled. Sulphide ore is estimated at 10.5 tons per truck load; siliceous sulphide 9.5 tons; schist and slag 8 tons; quartz and porphyry 7.5 tons, and diorite, greenstone, oxide, talc, and dike rock at 7 tons. All tramming trucks have been especially

reenforced to stand up under the heavy loads. They are equipped with steel cabs, hood and radiator protectors. The 6-wheel trucks (fig. 13) have compound transmissions and solid rubber tires. The tractor trucks are also equipped with compound transmissions and in addition have a high-speed reverse which is very useful, as a good deal of backing up is necessary.

Considerable experience has been derived from the trucking problem in the United Verde pit. So far the 6-wheel, 4-wheel-rear-drive trucks have proved to be the most satisfactory. A better distribution of the load is possible, and the flexible arrangement of the rear end protects the frame from undue strain. Good traction is also provided, and the truck is able to negotiate all roads, except on grades above 10 per cent and when the roads are extremely muddy. Rubber wear on the 4-wheel trucks has been found to be much greater than on the 6-wheel trucks. The tractor trucks are useful only for keeping operations going under extremely bad conditions, such as in deep mud and up steep grades. They are especially valuable during bad weather, but can not compare with the 6-wheel trucks under fair conditions.

The roads in the United Verde pit are bad. In fact, the levels are kept as clean as possible so that a truck can run anywhere. The grades are steep, uneven, and frequent. The heavy loads handled and conditions due to occasional bad weather subject the United Verde trucks to extremely severe service. Nevertheless, the situation is being met very successfully.

A truck repair man for making minor repairs is kept in the pit on each shift. For large repair jobs the truck is removed from the pit and put into the shop.

During the year 1928, 167 cubic yards or approximately 390 tons of material were trammed per truck shift. The average length of haul is 500 feet one way. Gasoline consumption amounted to 1 gallon for 11.69 cubic yards trammed. For each shovel shift 2.70 truck shifts were required to keep the shovels operating at full speed.

The miscellaneous light trucks are used for various service around the pit. The 5 to 6 ton trucks are used for handling drill steel, churn-drill bits, casing, and heavy material. The 3½-ton truck is equipped with an acetylene burning outfit and is used for transporting tools and parts for repair work in the pit. The 2-ton truck hauls supplies from the warehouse and the 500-level shops to the pit, as well as transporting the dump crew to Hopewell when it becomes necessary to shift the track on the dumps. The light 1-ton truck delivers powder and blasting supplies where needed in the pit, as well as drilling material.

The gasoline-powered crane is used in doing repair work in the pit as well as for handling heavy material; as it is equipped with a ½-cubic yard clamshell bucket it can be conveniently used for small excavating work in the pit and elsewhere.

The road scraper assists in keeping the roads clean in the pit and around the mine plant.

One of the 85-ton locomotives, 10 of the 25-cubic yard air-dump cars, and one of the 4-yard steam shovels from the major stripping operations are retained for reclaiming material placed on the dumps during this first work.

Miscellaneous shops, a transformer house, gasoline storage tanks, and a change room have been erected adjacent to the pit on the 160 level.

Drilling and Blasting

General. - In the lower pit care must be taken to break the material clean to avoid dilution. The depth, spacing, and location of holes are dependent entirely on the character of the material to be blasted. Three types of drills are used - jackhammers, a drifter mounted on a tripod, and electric traction churn drills. In breaking ground along contacts, in inaccessible places, and in bank-trimming the air drills are used; however, in uniform material and in hot ground the churn drills are preferred. In the lower pit, actual burning sulphides in the underground fire zone were encountered, which presented a difficult blasting problem.

A comparison made on the cost of breaking ground by the use of air drills and churn drills showed that although the cost of powder consumed in blasting churn-drill holes was 50 per cent more, a considerable saving was made in labor, power, and repairs. The total cost per ton broken with the churn drill was found to be 16 per cent less than with the air drill.

Air Drills. - Jackhammers are used to drill toe holes and vertical holes. Methods and equipment are similar to those employed in the major stripping operations, except that the jackhammer is an improved type.

The heavy drifter on a tripod mounting is used only for drilling toe holes in the very hard sulphide which can not be drilled with the jackhammers.

Air-drill holes are loaded with 35 or 50 per cent strength gelatin dynamite. The charges are fired with No. 8 electric detonators from the 440-volt line. The use of the powder-loading machine was abandoned, partly on account of irregularity in spacing and chambering of holes, but largely because of the hot, steaming ground encountered, which made the use of the granular powder unsatisfactory.

No "dobie" blasting is done in the pit. All bowlders are block-holed. The cost of bock-holing is about 2 cents per ton of material broken.

Churn Drills. - Four electric-powered churn drills are used in the United Verde pit in place of the old steam drills. Fuel cost on the steam drills amounted to $8 a shift against a power cost of $0.60 a shift on the new drills. Other improvements such as a wire-rope drilling cable favored the use of the new drills. A churn-drill crew consists of a driller and helper.

Churn-drill holes are drilled on the edge of the bank. Drilling speeds vary from three-fourths of a foot to 8 feet per hour. Holes are drilled 5 feet more in depth than the height of the bank. If the burden is too great, machine-drill toe holes are used with the churn-drill holes. Holes are spaced according to the conditions but are generally 10 to 12 feet apart. However, in hot ground, where it is impossible to place the desired amounts of explosive in the hole, the holes are spaced 4 to 5 feet apart. Hot and gassy ground is generally drilled with churn drills, as the larger hole can be more quickly and safely loaded with a greater amount of gelatin. Also fewer men have to be kept on the bad ground.

As a rule between 150 and 250 pounds of 35 per cent or 50 per cent straight gelatin dynamite are used in a hole. Some granular ammonia dynamite is used where it can be kept dry. Holes are generally tamped with sand. Blasting switches are located conveniently on the upper levels, and the blasting wires are wound on a reel adjacent to the switch. From these reels the blasting wires can be run out to any location in the pit.

Liquid oxygen explosive was experimented with in blasting churn-drill holes. It was found to effect a considerable saving, but because of insufficient knowledge of the action of "LOX" in possible hot ground, it was decided to be unsafe to continue its use.

Blasting Hot Ground. - In holes where heat is encountered a potentiometer with an iron-constantin thermocouple is used to measure the temperature. The highest temperature that has been measured in a hole is 780°F. A temperature of 120°F. is considered safe for loading and tamping a hole, using the ordinary methods. If cooling a hole is possible, the cooling is accomplished by running water into the hole for a period of 1 to 24 hours, or by using sand. By dropping sand in the bottom of the hole, the heat has been reduced in several instances from 212 to 120°F. long enough to permit loading and firing. When sand is used the hole must be chambered sufficiently to hold both sand and explosives. Sand is placed in the hole with a small amount of water, which washes the sand into the crevices around the chamber. The hot gasses are sealed off and the sand does not become heated readily as it is a poor conductor of heat. As many as three wheelbarrows of sand have been placed in some holes.

For holes that can not be cooled, a torpedo is made which consists of an insulating cartridge filled with explosives (fig. 14). The torpedo used in small machine-drill holes consists of an ordinary mailing tube filled with gelatin dynamite. For the larger 6-inch churn-drill holes a large paper tube with a 4-inch inner diameter and ½-inch wall, 6 to 8 feet long, is used. A wooden plug is driven in one end and a wire by which the tube can be lowered into the hole is fastened to it inside the tube. The charge and detonators are placed in the tube and covered with sand. In shooting a round of hot holes the torpedoes are wired and connected with the blasting line and placed off of the ground adjacent to the holes. When the blasting signal is given, the torpedoes are immediately lowered into the holes so that they can be detonated as soon as the men are in the clear. During the short interval that the torpedoes are in the hot ground the heat is not able to penetrate the tube. These hot holes are not chambered nor is tamping used

Thirty-five per cent gelatin dynamite and some granular ammonia dynamite are used in the torpedoes. Care must be taken that the granulated powder does not leak out of the torpedo.

Hot ground can not be blasted efficiently, as enough powder can not be put in the torpedo and what is in the torpedo is not in a chamber nor tamped. The use of the small torpedoes for the machine-drill holes is particularly inefficient. However, it is necessary to blast the hot ground in this manner, and a fair degree of success is being obtained. Accidents caused by blasting hot ground have been eliminated by the use of the torpedo.

Pit Operation

At present (November, 1929) 21,000 tons of direct-smelting ore and 3,500 tons of converter ore are mined weekly in the pit, and milling ore is mined as called for. In addition, the low-grade material which is placed on dumps at the mouth of Hopewell Tunnel is handled. It is estimated that 1 ton of low-grade material should be removed from above the No. 5 pit level for each ton of direct-smelting ore mined in order to make the ore available below this level.

Every morning the pit foreman is informed from the smelter and by the foreman of the tunnel transportation what the pit requirements for the 24 hours will be. This information is passed on to the other shifts so that the order can be completed. At the beginning of each shift the crews are told to what shovel, level, and place they are consigned, and are moved about during the shift if any changes arise. Truck drivers and dumpers are assigned to certain shovel crews and raises. Enough trucks should be available to keep the shovel busy at all times. On a short haul two trucks are sufficient, but as many as six trucks are necessary for some of the longer hauls. The truck dumper is stationed at the raise. It is his duty to keep the dump clean, adjust the safety chains on the trucks, see that the dump body is properly lubricated, and keep a record of the loads dumped.

During the shift the pit foreman is kept informed by the tunnel foreman as to the condition of the raises. Close cooperation has to be maintained in order to prevent the material from hanging up in the raises. The material should not be allowed to pile up in the raises above the grizzlies, as the larger pieces are likely to form an arch and the fine material often cements. Also if a heavy rain occurs and the material in the raise becomes muddy, a tremendous hydrostatic pressure may be developed. In one instance the pressure on being released forced out the chutes of the ore bin and flooded the haulage tunnel.

Clogging of raises with large rocks has been almost eliminated by not allowing any piece to be loaded which will not pass through the shovel dipper. However, the larger pieces are often held up on the grizzlies and the material starts to back up in the raise before they can be broken. An attempt is made to mix the coarser material with the finer in order to prevent choking the raises by the cementing and packing of the finer material.

The transfer raises are accessible on the mine levels, but are sealed off by doors and bulkheads from the mine workings. In case a raise becomes clogged, all dumping into that raise is stopped as soon as the condition is discovered. The raise is then inspected on various levels to determine where the choking occurred. Generally the material can be loosened by blasting in the raise. At times it has been necessary to tie as many as three or four 20-foot tamping sticks together with a charge placed on the end, and push it up the raise where the material is hung-up. On several occasions material has been loosened by allowing water to run down the raise which washes out the finer material wedged between the larger rocks.

At present eight shovel crews are maintained, three each on the day and afternoon shifts and two on the night shift. A spare shovel is available during the day for the repair crew to work on.

During 1928 the locomotive crane equipped with a clamshell bucket was used on the 160-level bench below the high diorite bank for cleaning up the material that slid off the bank. The loaded cars were dumped into the main mine waste raise. During the year 14,886 cubic yards was removed by the crane on this bench.

The day shift begins at 8 a.m., the afternoon shift at 4 p.m., and the night shift at midnight. Blasting is confined to regular hours - noon, 4 p.m., 8 p.m., and midnight - so that there will be a minimum of delay.

Sampling

A sampler who works on day shift samples the banks each day after they are cleaned up. The assay results of the samples are given to the pit foreman and powder foreman who plan the work accordingly. Drill holes are occasionally sampled but generally the class of material can be accurately determined from a knowledge of the geology and the value of the face of the bank. Samples are also taken from the broken material ahead of the shovel by the assistant samplers; one sample is taken for each 10 trucks loaded. A record is kept of the material removed. The bank sample assays are mapped for use in estimating the character of the material on the lower benches.

Disposal of Material

After the material is dropped into the raises it is hauled out of the tunnel and dumped into transfer bins at the entrance of Hopewell Tunnel. There the direct-smelting, converter, and concentrating ores are transferred to large cars to be hauled to the smelter. Low-grade material is placed on various dumps at Hopewell; separate dumps are maintained for leaching, low-grade converter, low-grade sulphide, and waste. Most of the waste from the pit is dumped into a waste transfer raise from which it can be distributed through the underground mine for stope filling. However, with the increase in production it was necessary to place some waste on the dump in order to make the ore available.

Classification of Pit Material

Material in the United Verde pit is classified according to the following schedule:

I. Converter Ore.

A. Smelter shipments.

1. Aluminous converter material, such as quartz porphyry to contain a minimum of 60 per cent silica. Alumina present must not exceed one part to 5.5 parts silica. Values to exceed $4 a ton.

2. Oxide and quartz converter material to contain a minimum of 50 per cent silica. Values to exceed $4 a ton.

3. A small amount of sulphide material will be permitted if silica content exceeds 65 per cent silica and values $4 a ton.

B. Hopewell stock-pile shipments (Dump No. 34).

1. All converter material with value less than $4 a ton.

II. Direct-Smelting Ore.

A. All sulphide material containing a minimum of 1.0 per cent copper. Basic and siliceous sulphide material to be mixed so that the average silica content of the ore received at the smelter is about 23 per cent.

B. Low silica content oxide and old filling containing a minimum of 2.0 per cent copper, or 1.5 per cent copper when gold and silver value is in excess of $1 a ton, or no copper value when gold and silver value is more than $4 a ton may be added to the sulphide ores, provided that the mixture does not exceed 23.0 per cent silica, nor more than 1 part alumina to 5.5 parts silica, when received at the smelter.

III. Concentrating Ore.

A. Clean schist or porphyry having 1.0 per cent copper or better. Low-grade ore should be mixed with high-grade ore so that the resultant mixture will approximate 3.0 per cent copper. Under no circumstances will filling be mixed with the concentrating ore.

B. A small amount of clean siliceous sulphide with copper values between 1.0 and 5.0 per cent may be mixed with the schist and porphyry.

IV. Leaching Material (Dump No. 36).

A. All low-grade copper-bearing material between 2.0 and 0.7 per cent copper, excepting material suitable for smelting, milling, and the converters.

V. Low-Grade Sulphide (Dump No. 37).

A. All sulphide material below 1.0 per cent copper not included in the above.

VI. Waste.

A. All barren material.

As no bedding bins exist in the Clarkdale smelter it is necessary for the pit department to control the silica and alumina content of the direct-smelting ore. The direct-smelting ore mined in the underground mine also has to be taken into consideration when determining the proper mixture. It is therefore necessary for the pit to determine the proportion of basic, siliceous sulphide, and stope filling which enters into the smelter mixture. The mixture that arrives at the smelter should contain about 23 to 24 per cent silica. If the mixture is off one day, an attempt is made as soon as possible to correct it by adding more basic or siliceous sulphide ore, as the need may be.

The United Verde pit differs frommost pit operations in that it is producing a high-grade ore. The direct-smelting ore mined averages about 5 per cent copper, while the concentrating ore averages approximately 3 per cent and the converter ore 2.5 per cent copper.

Probably half of the stope filling encountered is shipped to the smelter as ore. The copper in the stope filling is due to the following causes:

1. In the early days of the mine, low-grade material which was not commercial at the time was thrown in as filling, especially low-grade schist which is hard to smelt.

2. Early mining methods were not clean. In several of the exposed stopes enough high-grade ore is scattered through the filling to make it all run as ore at the present day.

3. Enrichment has resulted from deposition of copper sulphate and copper carbonate by water descending from the old workings. Enrichment has also been caused by gases ascending from the fire area and depositing copper salts in the filling.

4. Some of the sulphide in the fire areas has also been enriched by the removal of sulphur by roasting.

Dilution

An attempt has been made to estimate the percentage of dilution of the pit ore. The problem of estimating dilution is difficult, however, because unknown areas of ore are uncovered and because a large portion of the old stope filling is being shipped as ore.

A summary of the United Verde lower-pit operations for 1928 follows:

Table 3. - Summary of costs

United Verde Lower Pit Operations — Year 1928

Cubic yards stripping loaded	322,538
Cubic yards ore loaded	276,055
Total cubic yards loaded	598,593
Dry tons ore loaded	731,488

Direct operating costs per cubic yard based upon total yardage (ore and stripping)

	Labor	Comp. air[1] drills and steel	Power cost	Explo-sives	Repairs[2]	Supplies[3]	Miscel-laneous	Total
Shovel operation	$.047	- -	$.012	- -	$.117	$.001	$.003	$.180
Drilling and blasting	.128	$.084	.002	$.082	- -	.027	.004	.327
Truck tramming	.055	- -	- -	- -	.113	.019	- -	.187
Total operation.	$.230	$.084	$.014	$.082	$.230	$.047	$.007	$.694

1 Includes repairs to drills and sharpening steel and drill bits.
2 Includes labor, supplies, and power, used in repairing.
3 Includes gasoline and oil used on trucks.

The direct costs of shovel operation, drilling and blasting, and truck tramming are divided against stripping and mining in proportion to the tonnages of the two classes. The direct costs for stripping and mining for 1928 are tabulated below:

Direct costs for stripping and mining, 1928

	Per cubic yard	Per ton
Stripping (322,538 cu. yds.):		
Shovel operation	$.161	- -
Drilling and blasting	.293	- -
Truck tramming	.160	- -
Dump expense	.038	- -
Miscellaneous	.018	- -
Total	$.670	- -
Mining (276,055 cu. yds. or 731,488 tons)		
Shovel operation	$.204	$.077
Drilling and blasting	.368	.139
Truck tramming	.219	.086
Total	.791	.302

General data for 1928 are tabulated as follows:

Total 8-hour shovel shifts	1,285.75
Total 8-hour truck shifts	3,471.13
Total 8-hour churn-drill shifts	1,384.83
Total churn-drill footage	37,812
Feet per shift	27.3
Pounds of explosives per cubic yard	0.483
Total man-shifts	48,374
Cubic yards per man-shift (total material)	13.21
Tons per man-shift (total material)	32.62
Labor cost per cubic yard	$.3678
Operating supplies per cubic yard	$.1137
Repair supplies per cubic yard	$.1291
Total supplies per cubic yard	$.2428
Total cost of repairs per cubic yard	$.2728

RECLAIMING DUMPS

As converter ore is not abundant in the lower pit it is necessary to ship some of the converter material from the dumps which were established at the time of the major stripping operations. The material is loaded by a standard 4-yard shovel directly into cars which transport the material to Clarkdale. This is not a continuous operation, but the shovel is steamed up and the material loaded when called for. The cost for loading 39,772 cubic yards or 67,175 tons from the dumps for the year 1928 is tabulated as follows:

	Per cubic yard	Per ton
Shovel operation	$0.093	- -
Locomotive operation	0.074	- -
Drilling and blasting	0.041	- -
Miscellaneous	0.084	- -
Total	$0.292	$0.173

LEACHING

Water from the mine is run through the dump of leaching ore and takes some of the copper in the low-grade material into solution. The solution is gathered in ditches at the bottom of the dump and run over scrap iron in precipitating troughs, where the copper is recovered. The copper precipitate recovered contains about 80 per cent copper.

LABOR

In November, 1929, 225 men were employed in the United Verde pit. At present they are paid a 15 per cent bonus above a base scale. This bonus is based on the price of copper and on general operating conditions.

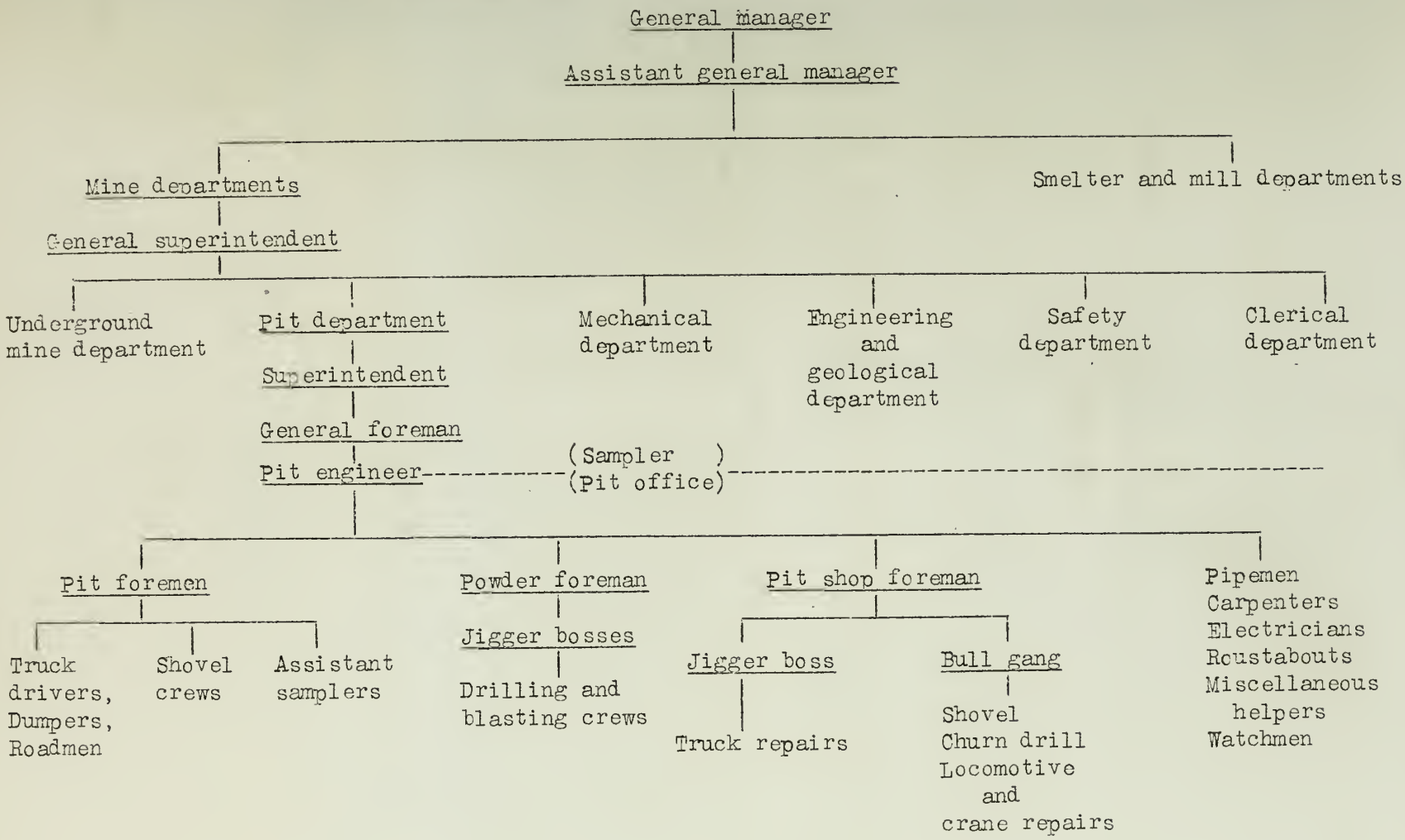
An efficiency bonus based on the work performed is also paid for most classes of labor. Standards are set for each class of work and a record is kept of the performance of each man or group of men. The men are paid on a basis of one-half of the work performed in excess of the standard amount. For instance, if the efficiency is 150 per cent, the workman receives 25 per cent of his base rate as a bonus. In some cases the bonus is figured on the basis individual work; in others the performance of a group is considered.

ADMINISTRATIVE ORGANIZATION

The general form of organization of the United Verde Copper Co., with more detailed form of the pit organization, is given in Table 4.

Cooperation between the mine departments, as well as the smelter departments, is maintained by the pit superintendent. An engineer is employed to supervise the engineering, sampling, and clerical work in the pit. Assistance in engineering and geological work is furnished by the main office. Pit records are kept in the pit office. The pit office also assists the main time office and safety department.

TABLE 4.- Administrative organization of the United Verde open-pit mine



SAFETY

Safety work in the pit is supervised by the safety engineer in charge of the entire Jerome operations. Hard-toe shoes are required, and goggles must be worn when performing certain classes of work. Close co-operation has to be maintained between the working sections of the mine, especially in the matters of blasting around raises and allowing men to be in underground workings which might be disturbed by the pit operations.

During 1928 five lost-time accidents occurred, resulting in a loss of 101 shifts, or of 1 shift for each 479 shifts worked.

MINE FIRE

An enormous amount of sulphur gases and smoke from the underground fire was encountered as the ground was dug in advancing the first bench below the 160-foot level on the west side of the pit. Respirators, gas masks, and extra crews were tried without success. Part of the pit had to be abandoned for a while.

In some areas the actual sulphide ore was burning; in others only the timbers started burning when exposed to the air. The timbers in some of the old stopes were completely burnt out, and the filling had cemented and formed casts where the timbers had been.

In the early part of 1927 it was decided to use slimes to seal off the fires. Tailings from the mill at Clarkdale were used with immediate success; no part of the pit is now uncomfortable to work in. The tailings were poured on the surface and pumped into churn-drill holes in the fire areas, thus filling the cracks and crevices from which the gas and smoke escaped and sealing off the air. In order to eliminate the high cost of transporting the tailings up the steep grade from Clarkdale, a sliming mill was constructed adjacent to the pit. A drift was run into the quartz porphyry and a glory hole started to obtain material. The slime plant consisted of a jaw crusher, rolls, ball mill, and storage tanks. The capacity of the mill was approximately 65 tons solid per shift. The slime consisted of 50 per cent moisture. The solids were ground so that about 35 per cent was minus 100 mesh.

The slimes were later pumped into the underground fire areas through the bulkheads which sealed off the fire country and assisted materially in reducing the mine fire.

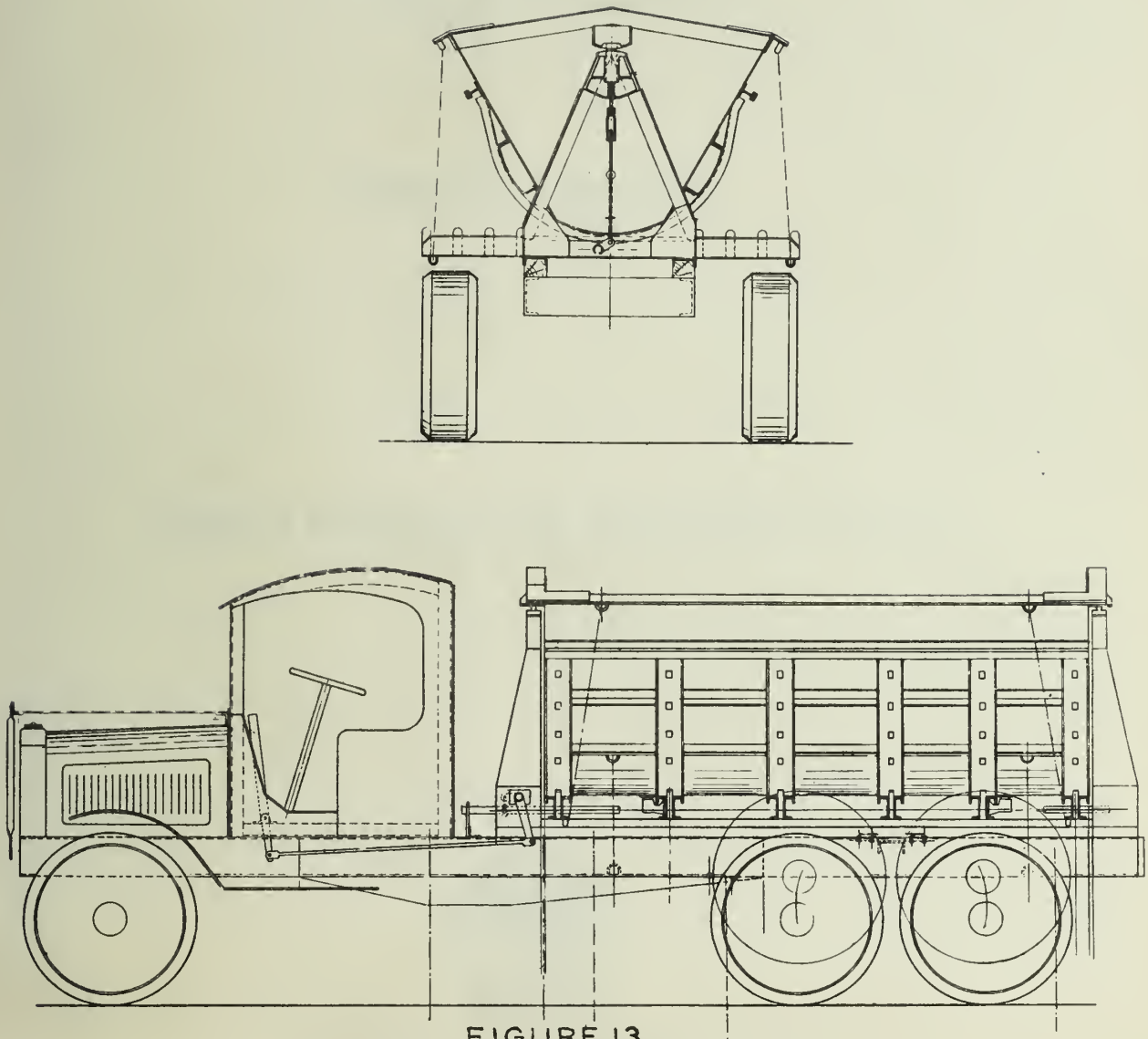


FIGURE 13.
5-CUBIC YARD 6-WHEEL TRUCK

I. C. 6249 March, 1930

DEPARTMENT OF COMMERCE
UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

WORLD RESERVES AND RESOURCES OF TIN

BY

CHAS. W. MERRILL

I.C.6249.
March, 1930.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

WORLD RESERVES AND RESOURCES OF TIN[1]

By Charles White Merrill[2]

INTRODUCTION

Tin has become more and more indispensable to American industry in recent years. The canning and automobile industries, the outstanding consumers of tin and its alloys, have grown so great that over 40 per cent of the world's virgin tin production is consumed in the United States. Over 99 per cent of the virgin tin we require is imported from foreign sources. Domestic production of virgin tin has always been so small as to be negligible, and long-continued search has failed to reveal tin resources which will be available in the future at present or considerably higher prices. Only by the recovery of tin from scrap has the United States succeeded in supplying any appreciable portion of its requirements.

It is apparently inevitable that the United States will remain dependent upon foreign production for its future supply of virgin tin. For this reason the tin resources[3] of the world, including the tin reserves of producing companies, are of very vital interest to American tin consumers. Upon knowledge of resources and reserves must depend the stabilizing of metal prices. Recently the price of tin has fluctuated from a high of 72.50 cents in 1926 to a low of 38.37 cents in 1929. The former excessive price was made possible by the fear of shortage on the part of American consumers.

Though it is thus obvious that estimates of future trends of production and price must be based upon knowledge of world resources and reserves, precise data for compilation of world totals are not available. The Chinese placer miners of Malaya who own large tin-bearing tracts do not find it necessary to measure their reserves; the Dutch Government refuses to disclose the reserves of the Banka placers; and the lode mines, from which one-third of the world production is derived, do not generally attempt any extensive development of reserves beyond their relatively immediate requirements.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6249."

2 - Mining engineer, common metals division, U. S. Bureau of Mines.

3 - In studies of the common metals division of the Bureau of Mines the following distinction is made: Reserves include merchantable ore actually demonstrated by exposure and precise measurement.
Resources include reserves as defined plus expectancies of ore based upon geological evidence or upon the possibility of future utilization of mineralized material not workable as ores at present prices.

Nevertheless, it has been deemed probable that the tin-consuming industries of the United States would welcome an attempt to make such compilations of reserves as the avail able data permit and to indicate as far as possible the nature of unmeasured or unrevealed reserves.

PUBLISHED RESERVES

Published data give the reserves covering only one-fourth of the world production. These reserves are shown in the following table.

TABLE OF PRODUCTION AND RESERVES OF THE PRINCIPAL TIN-MINING COUNTRIES OF THE WORLD

Country	Total production in 1928, long tons	Production from companies reporting reserves				Reserves as reported by producing companies		Life based on reserves and 1928 production		Reserves reported by nonproducers	
		Placer, long tons	Lode, long tons	Total, long tons	Per cent of total production	Placer, long tons	Lode, long tons	Placer, years	Lode, years	Placer, long tons	Lode, long tons
Australia	2,900	211	-	211	7	2,639	-	12½	-	3,764	7,000
Belgian Congo	1,200	-	-	-	-	-	-	-	-	-	-
Bolivia	[1]41,409	-	23,548	23,548	57	-	130,000	-	5½	-	-
China	[1]7,033	-	-	-	-	-	-	-	-	-	-
United Kingdom	2,782	-	595	595	21	-	1,939	-	3	-	1,050
India	2,200	1,048	-	1,048	48	12,967	-	16½	-	8,192	-
Japan	(2)	-	148	148	-	-	3,729	-	25	-	-
Federated Malay States	[1]61,935	8,057	1,777	9,834	16	225,580	3,555	28	2	170,806	7,000
Unfederated Malay States	[1]2,435	-	-	-	-	-	-	-	-	-	-
Netherland East Indies	36,600	-	-	-	-	-	-	-	-	-	-
Nigeria	9,100	6,658	-	6,658	73	90,553	-	13½	-	13,500	-
Siam	7,900	1,299	-	1,299	16	35,056	-	27	-	40,328	-
Union of South Africa	1,257	-	734	734	58	-	1,783	-	2½	-	-
Others	[3]2,949	-	-	-	-	-	-	-	-	1,343	-
Total	179,000	17,273	26,802	44,075	25	366,795	141,006	21	5¼	237,933	15,050

1 - Exports.
2 - Production included with "others."
3 - Estimated.

CHARACTER OF TIN RESERVES AS RELATED TO METHODS OF MINING

The commercial tin deposits of the world may be divided into two main groups: alluvial, here including residual deposits and tailings, as well as stream deposits; and lode or vein deposits. During 1928 approximately two-thirds of the production came from alluvial deposits and the remaining one-third from lode mines.

The methods of extracting the tin concentrates have an important influence on the data available concerning the reserves. Placer deposits worked by hand-washing methods are seldom tested to prove the reserves because the low cost and portability of equipment make such testing an unnecessary expense. On the other hand, when a tract of alluvium is to be worked by dredging, the cost of erecting the equipment and of preparing the deposit for exploitation is so great that sound business demands a proved reserve which will amortize the initial capital outlay and pay a profit during operation. The useful life of the average dredge is ordinarily placed between 15 and 20 years. It follows that most companies which extract tin concentrates by dredging have data concerning their reserves which usually assure a life of a decade or more from the date of initial production. Between the extremes of extraction by hand-washing and extraction by dredging there are numerous other methods such as hand-jigging, ground-sluicing, small-scale sluicing, gravel-pumping with sluicing, and steam-shovel operations. These methods require varying degrees of initial capital expenditure and ordinarily are undertaken only after enough reserves are proved to amortize the initial capital outlay.

Lode deposit mines in general have smaller proved reserves than placer mines because of the greater expense of blocking out lode ore compared with that of drilling alluvial deposits. Although few tin lode mines have proved reserves which would assure current production for more than four or five years, it may safely be assumed that the tin resources of lode mines taken as a whole are far in excess of any such proved reserves. In fact, it appears most probable that consumers of tin must look to lode mines for their supplies in the not very distant future, much as the consumers of gold look less and less to the rich placers as source of the precious metal.

The course of the tin-mining industry shows a striking resemblance to that of gold mining except that gold mining has advanced much farther in the transition from placer and bonanza lode mining to the large unit treatment of low-grade lode deposits. The great age and depth of the Cornish tin mines which, though productive for centuries, still supply between 1 and 2 per cent of the world's tin, is apt to be paralleled, at least to some degree, by other tin lode deposits of the world.

DISCUSSION OF THE PLACER AND LODE RESERVES OF THE PRINCIPAL TIN-MINING COUNTRIES OF THE WORLD

Federated Malay States

In 1928 the Federated Malay States was the largest producer of tin in the world. The production of 61,935 long tons may be classified as follows: 31,584 tons, or 51 per

cent, was derived from Chinese-operated mines; 18,750 tons, or 30 per cent, from European-operated dredges; 8,155 tons, or 14 per cent, from placer operations other than dredging conducted by Europeans; and 3,406 tons, or 5 per cent, from lode mines. For the properties held by Chinese producers, from which more than half the production was derived, no figures of reserves are available. It is probable that a large proportion of this production comes from deposits which have never been tested, as much of the tin is won by methods which require practically no capital for equipment. So far the Chinese have not used dredges but are reported to have made increasing expenditures for other modern equipment. Nothing more than a guess can be made of their reserves, but undoubtedly they are sufficient for some time to come.

Out of a total dredge production of 18,750 tons, 7,866 tons came from properties reporting reserves. These reserves if exploited at the 1928 rate assure a life of 28 years to the group of reporting companies without additional developments or discoveries. Many of the dredging companies not reporting reserves are among the older producers. Some of them are undoubtedly nearly worked out both in respect to ore and serviceability of equipment; but others are developing further reserves, acquiring additional tracts, and prolonging their expectancy more or less indefinitely. In addition to the operating areas there are alluvial tracts already proved to contain over 170,000 tons of tin available at present prices. Most of this proved ground will go into production within a year or two unless new factors arise to stop developments now in progress. Since most of this reserve is to be exploited by dredging it should require at least 20 years to work out and should contribute some 5,000 to 10,000 tons of tin a year in the near future.

There is little available reserve data on the European-controlled placer operations other than dredging, but it is probable that reserves held by such companies afford a shorter expectancy than do the reserves held by dredging companies.

Of the 3,406 tons of tin derived during 1928 from lode mines, 1,777 tons came from mines reporting reserves that assure two years of operation at the 1928 rate. These reserves will undoubtedly be increased as mining and development proceed. Seven thousand tons of tin have been blocked out in lode ore bodies which have not yet been brought into production.

The lack of data on reserves of the Chinese-owned mines makes a forecast of the future production of the Federated Malay States exceedingly difficult. A Government official of the Federated Malay States, though asking not to be directly quoted, made the following statement to Frank L. Hess[4] chief engineer, rare metals and nonmetals division of the United States Bureau of Mines, who has recently returned from a trip through some of the tin-producing areas of Asia: The output of tin in the Federated Malay States probably will gradually rise for the next five years, when it will reach a maximum and remain fairly steady for 20 years longer, after which a decline will begin.

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4 - The author wishes to acknowledge his indebtedness to Mr. Hess for much of the material contained in the discussion of the Far Eastern areas.

Many of the placer gold regions of the past have become the lode gold regions of the present, and it might be inferred that British Malaya had the prospect of becoming a great lode-tin producing area in the future. The geology of this peninsula, however, does not confirm such an inference. The major part of the tin appears to be derived from extremely low-grade, cassiterite-bearing stock works in schists, granites, and limestones. Only through the agency of placer concentration has the tin content been raised to a mineable grade. Large pieces of cassiterite are so rare that they are exhibited in the museums of Malaya. This apparently precludes the possibility that rich lodes exist, which will take the place of the placer deposits when the production of the latter declines. There also seems to be little likelihood that large undecomposed stockworks of mineable grade will be uncovered.

The British Government has an unusual degree of control over the tin industry in the Federated Malay States and in the Straits Settlements where the concentrates are smelted. It is therefore possible that the tin reserves of this area may be conserved or more rapidly exploited as a result of Governmental action. The recent decision to withhold public mineral lands from private entry and the inauguration of the six-day week in the tin fields indicate a conservation policy on the part of the present Government officials.

Bolivia

During 1928 Bolivia contributed 41,409 tons of tin to world production. Practically all this tin was extracted from lode deposits. Over one-half of the production, or 23,548 tons, came from mines for which reserve figures are available. These reserves assure the mines reporting them an average life of five and a half years at the present rate of production. No nonproducing properties report ore blocked out. Though five and a half years' reserves would amount to very little in alluvial areas, it indicates a very healthy condition when reported for lode deposits. Bolivia's great tin-bearing lodes might well become the backbone of the world's future tin production.

Netherland East Indies

No definite figures are available on the reserves of the Netherland East Indies, though its production of 36,600 tons of tin in 1928 placed it third among world tin producers. The largest producer is the Government, which operates chiefly on the island of Banka. Much of the tin is won by various open-pit methods which do not require the proof of reserves before exploitation. The other two chief producers are the Billiton Co. and the Singkep Co. which operate on islands for which their respective companies are named. Their operations, like those of the Government areas, have required little development of reserves. In 1925 W. R. Jones[5] said:

> The Committee, for example, which recently investigated the tin resources of Billiton for the Dutch Government and the Billiton

5 - Jones W R , Tinfields of the World London 1925, p 231.

> Company, reported that the proved areas of the island were sufficient to ensure an annual output of from . . . . . 7,000 to 8,000 tons, for a period of 10 years only; and it is within the knowledge of the author, who has visited Billiton, that extensive prospecting all over the island has not resulted in the discovery of important additional areas. The extent and depth of secondary stanniferous deposits can be closely estimated by boring, as has been done in the Dutch East Indies, and it is an interesting and very significant fact that 10 or 12 years hence the bulk of the rich secondary deposits of the Dutch East Indies, of Siam, and of Lower Burma will have been exhausted.

The improvement of mining methods and the possible development of lode deposits in the Netherland East Indies may prolong the life of this area well beyond the 10 or 12 years of the high-grade alluvial areas. Whether production can be maintained at the present level and at near present costs only the future can tell.

Nigeria

Nigeria produced 9,100 tons of tin during 1928, almost wholly from alluvial deposits. Over two-thirds of this production, or 6,658 tons, came from mines which reported their reserves. The reserves as reported afford their companies an average life of 13½ years at the 1928 rate of production. Companies not yet among the producers reported reserves of 9,450 tons of tin. Most of the companies reporting reserves also reported large mining areas as yet untested and, in addition, prospecting rights over much larger areas. It seems reasonable to suppose that Nigeria will greatly extend the life of its tin production as development is carried forward.

Siam

Siam produced 7,900 tons of tin in 1928, of which 1,299 tons came from operations reporting reserves. These reserves were sufficient to assure their owners of production at the present rate for 27 years. The reserves of nonproducers stood at the high figure of 40,328 tons, which would indicate a future expansion for Siam. The production of Siam is practically all from alluvial deposits, and most of the nonproducing reserves either have dredges under construction or are endeavoring to finance dredge operations.

China

China is estimated to have produced 7,033 tons of tin during 1928. Though there are no reserve figures available for the Chinese deposits, the fact that much of the production comes from lodes and residual material at the outcrops of lodes, leads to the belief that China's production of tin should increase rather than decrease for many years to come.

Australia

During 1928 Australia produced 2,900 tons of tin, 211 tons of which came from mines reporting reserves. The reserves of these producers assured an average life of 12½ years at the present rate of production. Nonproducers reported reserves of 7,000 tons of tin. The tin producing areas of Australia are widely scattered and include both lode and

placer deposits. There are great reserves of very low-grade disseminated tin ore which should become available with an advance in mining methods or an increase in the price of tin. Australia's reserves appear to be very large but under present mining and price conditions are only on the border line of availability.

United Kingdom

During 1928 the United Kingdom produced 2,782 tons of tin, of which 595 tons came from mines reporting reserves assuring a life of three years; Nonproducing properties reported reserves of 1,050 tons of tin. All production was from the tin-bearing region lying chiefly in Cornwall, which is a striking example of the persistence of tin production in areas depending on lode deposits. An important part of the 1928 production came from lodes whose discovery probably antedates the Christian era. Though only three years' supply of ore is blocked out, it is not at all improbable that this famed tin-yielding area will continue to produce important amounts for many years to come.

India

The province of Burma, India, produced 2,200 tons of tin during 1928. Of this production 1,048 tons came from properties reporting reserves. These reserves were great enough to assure their holders their present rate of production for 16½ years without further development. Properties not yet in production reported reserves of 8,192 tons. Practically all production is won from alluvial deposits located on or near the coast and near the southern extremity of the province.

Union of South Africa

During 1928 the Union of South Africa produced 1,257 tons of tin. Of this amount 764 tons came from mines reporting reserves sufficient for two and a half years' production at the 1928 rate. Most of the tin comes from lode mines, and it seems probable that further development of the Union will bring to light large unknown reserves and probably will not only prolong the period of production but also increase the average annual output.

Other Tin-Producing Countries

The Unfederated Malay States produced 2,435 tons in 1928, the Belgian Congo, 1,200 tons, and all other countries 2,949 tons; no reserve figures are available for their producing properties, but their nonproducers report a total of 8,011 tons of metallic tin in ore blocked out. Some of the countries included under this classification are new in the field of tin production, and it therefore seems probable that their total reserves when proved will be of considerable size in comparison with their present production.

Conclusions

One quarter of the tin production of the world is extracted by companies which report their tin-ore reserves. It is probable that the producing companies which do not

report reserves are not generally as well backed by reserves as are the reserve-reporting companies. It is true, however, that many of those not reporting reserves refrain from reporting because of a policy of business secrecy rather than from a desire to conceal an unfavorable state of affairs. This would apply particularly to such producers as the Dutch Government on the island of Banka, which produced 21,000 tons of tin, or 12 per cent of the world total, in 1928. There are some facts that stand out fairly clearly as regards tin reserves: The tin famine which has been predicted as imminent for the last few years undoubtedly has been postponed for some years by the flood of new capital which has entered the tin-mining industry during the recent years of high tin prices; all the important alluvial areas are increasing their outputs with each succeeding year; Bolivia, the greatest lode producer, is increasing its production by further development, better technology, and improved transportation; and the other lode producers, taken as a whole, are holding their own.

Fortunately for those who will consume tin in the future, the major tin-producing areas have been protected from excessive exploitation in the past by their location in the relatively less advanced and naturally more unfavorable regions of the world. The areas centering about British Malaya and including the Federated Malay States, Unfederated Malay States, the Netherland East Indies, Siam, and Burma, though well situated for cheap water transportation, are practically on the equator, in a region whose surface is heavily masked by dense tropical vegetation and peopled largely by industrially backward races. Both prospecting and development has progressed only by overcoming great difficulties in comparison with those encountered in producing most metals in Europe and the United States.

Bolivia, the second greatest tin area of the world, has been greatly handicapped by great altitude, rugged topography, lack of cheap local power, and very expensive transportation. The tin-bearing region of Nigeria has the difficulties attending the inland tropics (10°N); inadequate local power and transportation have made mining difficult. The other areas, which as a whole have about average conditions for mineral production, together produced less than 10 per cent of the world's total tin in 1928.

Fortunate also for future consumers of tin is the fact that many of the uses of tin are not destructive, or are only partly destructive. During 1928 about one-third of the United States' consumption was supplied by domestic secondary tin. Tin entering the alloys is largely recovered to be reused in new shapes after remelting. Practically all tin entering the pigment and silk trades is completely lost when the article containing the tin is finally discarded. That used in the manufacture of tin plate, though at present largely lost, is subject to partial recovery as soon as prices warrant the collection and treatment of discarded tin cans. At present, most of the tin plate clippings, resulting from can manufacture, yield their tin coating for further use.

The reserves of tin appear to be sufficient to supply the world's tin requirements for many years, provided these reserves are exploited and used with care, every effort is made to stimulate substitution when economically possible, and recovery of seoondary tin is carried on with care and thoroughness.



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